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Assessment of Industrial Hazardous Waste Practices in the Metal Smelting and Refining Industry. Volume II Primary and Secondary Nonferrous Smelting and Refining

Calspan Corp, Buffalo, N.Y.

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ASSESSMENT OF INDUSTRIAL HAZARDOUS WASTE PRACTICES IN THE METAL SMELTING AND REFINING INDUSTRY

Volume II

Primary and Secondary Nonferrous Smelting and Refining

This final report (SW-145c.2) describes work performed for the Federal solid waste management program under contract no. 68-01-2604 and is reproduced as received from the contractor

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This report covers primary and secondary metal smelting and refining industries. Characteristics of each industry sector, including plant locations, production capacities, and smelting and refining processes, have been identified and described. Land-disposal or stored residuals have been identified and characterized for physical and chemical properties. State, regional, and national estimates have been made of the total quantities of land-disposed or stored residuals and potentially hazardous constituents thereof. Current methods employed by the primary metals industry for the disposal or storage of process and pollution control residuals on land are described. Methods of residual treatment and disposal considered suitable for adequate health and environmental protection have been provided. Finally, the costs incurred by typical plants in each primary smelting and refining category for current and environmentally sound potentially hazardous residual disposal or storage on land have been estimated.

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ABSTRACT

Investigations of on-land disposal of process and pollution control residuals from the United States metal smelting and refining industry were conducted. This volume presents the results of studies of the U.S. primary and secondary non-ferrous smelting and refining industry. Characteristics of each industry sector, including plant locations, production capacities, and smelting and refining processes, have been identified and described.

Land-disposed or stored residuals, including slags, dusts, and sludges have been identified and characterized for physical and chemical properties. State, regional, and national estimates have been made of the total quantities of land-disposed or stored residuals and potentially hazardous constituents thereof.

Current methods employed by the primary metals industry for the disposal or storage of process and pollution control residuals on land are described. Principal methods include lagoon storage of sludges and open dumping of slags. Methods of residual treatment and disposal considered suitable for adequate health and environmental protection have been provided.

Finally, the costs incurred by typical plants in each primary smelting and refining category for current and environmentally sound potentially hazardous residual disposal or storage on land have been estimated.

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SECTION I

CONCLUSIONS

The primary and secondary non-ferrous smelting and refining industry disposes or stores large quantities of process and pollution control residues on land. These residuals are predominantly inorganic slags and sludges containing silicates, oxides and sulfates, and chlorides in some industries. Sludges are often residues of SO_2 scrubwater treatment or process wastewater treatment with lime. Consequently, they contain calcium sulfates, calcium sulfites, calcium hydroxides, and calcium carbonates. It was found that recycling of dusts from emissions controls was a relatively common practice, although some dusts are disposed on land. The high metallic content of dusts often allows their recycle to the production process, an economically attractive and viable alternative to land disposal.

The principal potentially hazardous constituents found in primary and secondary smelting residuals are heavy metals, including arsenic, cadmium, lead, zinc, copper, chromium, antimony, and nickel. Because of trace amounts in the concentrates and ores from which the metals are recovered, the primary base metal smelting and refining industries (i.e., lead, copper, zinc, antimony, mercury, tungsten, and tin) produce a wider variety of heavy metals in residues, including arsenic, cadmium, lead, zinc, copper, antimony, nickel and mercury.

Bauxite, the ore from which aluminum is recovered, is essentially devoid of toxic heavy metals. Fluorides and very small amounts of cyanide appear in sludge and potliner residues because of the fluoride contained in input cryolite (Na_3AlF_6), and cyanide produced in potliner consumption. Magnesium produced from electrolysis of seawater or from dolomitic limestone also uses heavy metal deficient raw materials and will have negligible concentrations of heavy metals in residuals.

The predominant practices used in the primary and secondary non-ferrous smelting and refining industries for disposal of residuals are lagooning and open dumping. Slags and other solid residues are generally open dumped on land. Scrubwater from wet emissions control and process wastewater with or without lime treatment is generally routed to unlined settling pits or to unlined lagoons. Settled sludge is often dredged from pits or lagoons and stored or disposed of on land. Industries which produce relatively small quantities of sludge will often leave sludges permanently in lagoons. The use of unlined settling pits and lagoons is the predominant practice.

The use of lined lagoons for sludge settling, chemical fixation of sludge before land disposal, and sealing of ground areas before hazardous waste disposition are techniques for adequate environmental protection. In a number of industries including primary copper, zinc and lead, immediate recycle or shipment of residuals to be further processed for metallics recovery (i.e., dusts, sludges) will preclude leaching of potentially hazardous heavy metal constituents.

Although the presence of potentially hazardous constituents in slags, sludges and dusts has been shown, it is generally not known if these materials are leached in disposal environments. In the event that significant leaching is demonstrated, a number of suggestions are made with respect to practices for adequate health and environmental protection. It is suspected that sludges and dusts may leach heavy metals and other potentially hazardous constituents to a greater extent than most slags because of fine particle size and consequent susceptibility to weathering processes. Solubility tests conducted on collected waste samples and described in Appendix B generally supported this hypothesis.

Recommended practices in the event of demonstrated significant leaching of potentially hazardous constituents include the use of lined lagoons for storage or permanent disposal of sludges. Leachable sludges which are dredged or pumped from lagoons or settling pits and dumped on land can often be chemically "fixed" so that leaching of heavy metals can be prevented. Alternatively, sealing of soil in disposal areas with bentonite or other sealants should prevent leachate percolation.

In the event that slags, calcine residues, retort residues, or other land-disposed or stored solid residues are shown to leach significantly, then soil sealing at disposal or storage areas would be needed. Collection of runoff from disposal dumps containing slags, sludges or dusts with leachable heavy metals or other potentially hazardous constituents may be needed. Collected runoff would require treatment before discharge or retention and evaporation in lagoons.

In a number of industry sectors, it was found that some sludges, dusts, or other residues are stored on open ground for periods ranging from months to years before processing for further metallic recovery. In such cases, immediate recycle or storage in concrete pits before reprocessing will preclude leaching of potentially hazardous constituents.

In some industries the use of dry air pollution control systems can greatly reduce or eliminate the quantity of land-disposed waste. Examples are the use of dry absorption beds in the primary aluminum industry and silica-impregnated baghouses in the secondary aluminum smelting sector.

Future air and water pollution controls are expected to increase the quantities of land-disposed sludges, particularly sulfite and sulfate sludge residues from control of SO_2 emissions from primary copper and secondary lead smelters.

SECTION II

PRIMARY NON-FERROUS SMELTING AND REFINING

This section presents the results of investigations and analyses of on-land disposal or storage of process and pollution control residuals from the United States primary non-ferrous smelting and refining industry. Characteristics of each industry sector including plant locations, production capacities and smelting and refining processes have been identified and described.

Land disposed or stored residuals including slags, dusts and sludges have been identified and characterized physically and chemically. State, regional and national estimates have been made of the total quantities of land disposed or stored residuals and potentially hazardous constituents thereof.

Current methods employed by the primary metals industry for the disposal or storage of process and pollution control residuals on land are described. Principal methods include lagoon storage of sludges and open dumping of slags. Methods of residual treatment and disposal considered suitable for adequate health and environmental protection have been provided. Finally, the costs incurred by typical plants in each primary smelting and refining category for current and environmentally sound residual disposal or storage on land have been estimated.

1.0 PRIMARY SMELTING AND REFINING OF COPPER (SIC 3331)

1.1 INDUSTRY CHARACTERIZATION

The major U.S. copper smelting and refining companies are vertically integrated and have mining, smelting, refining, fabricating facilities, and marketing organizations. Other large producers mine and process through the smelting or refining stage and other companies mine and concentrate their ore and ship the product to custom plants for smelting and refining.

Copper smelting capacity in the United States in 1974 totaled 8,024,000 MT of charged copper concentrate. As a broad generalization it takes 4 tons of copper concentrate to produce 1 ton of copper metal. Four companies control nearly 85 percent of the capacity. The capacity of electrolytic copper refineries in the United States in 1974 totaled about 2,412,000 MT or about 88 percent of copper metal output. Fire-refining capacities in 1974 totaled 334,000 MT or about 12 percent of the U.S. refinery capacity. The total quantity of refined primary copper produced in the United States in 1973 was 1,868,488 short tons and the average selling price was 59.5 cents/lb (Reference 2). Thus national sales are estimated as $2,223,500,000.

Eight companies operate 15 primary smelters and 11 companies operate 16 refineries. In 1972, the principal copper-producing states were Arizona (with 55 percent of the total), Utah (16 percent), New Mexico (10 percent), Montana (seven percent), Nevada (six percent) and Michigan (four percent). These six states accounted for 98 percent of the total domestic production. Smelters were generally located in the principal mining states with the exception of the El Paso, Texas, and Tacoma, Washington facilities. About half of the refinery capacity was located along the middle Atlantic coast with most of the remainder near primary smelters. Electrolytic refineries are located on the East Coast so as to be accessible to Eastern markets.

Output of copper at primary smelters in the U.S. in 1972 was 1.60 million MT (1.76 million short tons). Production of refined copper from electrolytic and fire refining was 1.70 million MT (1.87 million short tons), a record high. (Ref. 1.).

Table 1 gives the locations, capacities, process modes and other salient features of U.S. primary copper smelters and refiners. State by state, regional and national capacities for primary copper smelting and refining are given in Table 2.

The three major steps in producing copper metal are roasting, smelting, and refining.

TABLE 1 PRIMARY COPPER SMELTERS AND REFINERIES

Plant Name/Location	Smelting (Blister Copper) Tons/Year	Fire Refining Tons/Year	Elect. Refining Tons/Year
ASARCO/El Paso, Texas	100,000		
ASARCO/Hayden, Arizona	180,000		
ASARCO/Tacoma, Washington	100,000		120,000
Phelps-Dodge/Douglas, Arizona	4,100	142,000	
Kennecott/Hayden, Arizona	80,000		
Cities Service/Copperhill, Tenn.	15,000		
Anaconda/Anaconda, Mont.	202,000	202,000	
Kennecott/Hurley, N. Mexico	4,000	88,200	
Kennecott/McGill, Nevada	43,000		
Kennecott/Garfield, Utah	260,000	260,000	260,000
Magma/San Manuel, Arizona	183,000	183,000	57,000
White Pine/White Pine, Michigan	72,000	70,000	
Inspiration/Miami, Arizona	118,000	43,000	75,000
Phelps/Dodge/Ajo, Arizona	55,400		
ASARCO/Baltimore, Maryland			144,000
ASARCO/Perth Amboy, N.J.			160,000
Anaconda/Perth Amboy, N.J.			113,000
Anaconda/Great Falls, Montana			175,000
Kennecott/Baltimore, Maryland			178,000
Phelps-Dodge/El Paso, Texas			417,000
Phelps-Dodge/Mespeth, N.Y.			125,000

Table 2
GEOGRAPHICAL DISTRIBUTION OF PRIMARY COPPER SMELTERS AND REFINERS

		PLANT DISTRIBUTION BY CAPACITY (MT)			CAPACITY PER PROCESS (MT)		
STATE	NO.OF PLANTS	15 - 100K	100 - 300K	300 - 500K	SMELTING	FIRE REFINING	ELECTROLYTIC
ARIZONA	7	2	5	0	802,500	550,000	132,000
MARYLAND	2	0	2	0	0	0	322,000
MICHIGAN	1	1	0	0	72,000	70,000	0
MONTANA	2	0	2	0	202,000	202,000	175,000
NEVADA	1	1	0	0	43,000	0	0
NEW JERSEY	2	0	2	0	0	0	273,000
NEW MEXICO	1	1	0	0	4,000	88,200	0
NEW YORK	1	0	1	0	0	0	125,000
TENNESSEE	1	1	0	0	15,000	0	0
TEXAS	2	1	0	1	100,000	0	417,000
UTAH	1	0	1	0	260,000	260,000	260,000
WASHINGTON	1	1	0	0	100,000	0	0
EPA REGION							
I	0	0	0	0	0	0	0
II	3	0	3	0	0	0	398,000
III	2	0	2	0	0	0	322,000
IV	1	1	0	0	15,000	0	0
V	1	1	0	0	72,000	70,000	0
VI	3	2	0	1	104,000	88,200	417,000
VII	0	0	0	0	0	0	0
VIII	3	0	3	0	462,000	462,000	435,000
IX	8	3	5	0	845,500	550,000	132,000
X	1	1	0	0	100,000	0	0
NATION	22	8	13	1	1,598,500	1,170,200	1,704,000

Roasting. The object of roasting copper sulfide ores and concentrates is to regulate the amount of sulfur so that the material can be efficiently smelted and to remove certain volatile impurities such as antimony, arsenic, and bismuth.

Smelting. Roasted and unroasted materials are smelted after mixing with suitable fluxes in reverberatory furnaces. In this reducing or neutral atmosphere, copper and sulfur form copper sulfide and iron sulfide. The combination of the two sulfides known as copper matte collects in the lower area of the furnace and is removed. Mattes commonly contain 40 to 45 percent copper. Impurities such as sulfur, antimony, arsenic, iron, and precious metals are in the matte. The remainder of the molten mass containing most of the other impurities floats on top of the matte and is drawn off and discarded as a slag.

Converting is the final stage in the process of smelting and is accomplished by blowing thin streams of air through the molten matte in a refractory-lined converter to oxidize the ferrous sulfide to sulfur dioxide, to eliminate the sulfur as a gas, and also to form a ferrous slag containing trace metal impurities. When the converter cycle is finished, the converter is tilted to discharge the lighter slag and then the relatively pure copper metal, referred to as blister copper, into ladles in which it is transferred to a holding furnace and then to a poling furnace for fire-refining or casting into anodes for electrolysis.

The blister copper produced by smelting is too impure for most applications and requires refining before use. It may contain silver and gold, and other elements such as arsenic, antimony, bismuth, lead, selenium, tellurium, and iron. Two methods are used for refining copper - fire refining and electrolysis.

Fire Refining. The fire-refining process employs oxidation, fluxing, and reduction. The molten metal is agitated with compressed air. Sulfur dioxide is liberated and some of the impurities form metallic oxides which combine with added silica to form slag. Sulfur, zinc, tin, and iron are almost entirely eliminated by oxidation. Lead, arsenic, and antimony can be removed by fluxing and skimming as a dross. Copper oxide in the melt is reduced to metal by inserting green wood poles below the bath surface (poling). Reducing gases formed by combustion of the pole convert the copper oxide in the bath to copper. If the original material does not contain sufficient gold or silver to warrant its recovery, or if a special purpose silver-containing copper is desired, the fire refined copper is cast directly into forms for industrial use. If it is of such a nature as to warrant the recovery of the precious metals, the fire refining is not carried to completion but only far enough to insure homogeneous anodes for subsequent electrolytic refining.

Electrolytic Refining. In the electrolytic refining step anodes and cathodes (thin copper starting sheets) are hung alternately in concrete electrolytic cells containing the electrolyte which is essentially a solution of copper sulfate and sulfuric acid. When current is applied, copper is dissolved from the anode and an equivalent amount of copper plates out of solution on the cathode. Such impurities as gold, silver, platinum-group metals, and the selenides and tellurides of metal fall to the bottom of the tank and form anode slime or mud. Arsenic, antimony, bismuth, and nickel enter the electrolyte. After the plating cycle is finished, the cathodes are removed from the tanks, melted, and cast into commercial refinery shapes. The copper produced has a minimum purity of 99.9 percent.

1.2 WASTE CHARACTERIZATION

This section contains descriptions of production technology at typical primary copper smelters and refiners and the resultant byproducts or wastes which are either recycled directly, shipped to other smelters for further metallic recovery, or disposed of on land. Estimates are given for the quantities of wastes and potentially hazardous constituents thereof which are disposed of on land either in lagoons or open dumps. Smelters with fire refining and electrolytic refiners are considered separately.

1.2.1 Process Description

Smelting and Fire Refining. The hypothetical copper smelting and fire refining plant is assumed to have an annual capacity of 100,000 MT (110,000 short tons) per year. Approximately half of the U.S. copper smelters are smaller and half larger than this figure, and the average output capacity of all plants is within 15 percent of this figure. Copper smelters are estimated to operate 350 days/year; the daily capacity is, therefore, 286 MT (314 short tons) per day.

Smelter feed material, although composed chiefly of copper sulfide concentrates derived from porphyry ores, also includes a variety of copper-bearing byproducts from lead and zinc smelters as well as diverse kinds of copper-bearing concentrates from nonporphyry ores. Hence, the smelter feed may range in chemical composition as follows: Copper, 25 to 35 pct; sulfur, 27 to 33 pct; iron, 20 to 30 pct; and combined gangue constituents, 10 to 20 pct. Further, typical smelter feeds frequently contain trace to small amounts of several or perhaps all of the following elements: thallium, bismuth, cadmium, tellurium, selenium, platinum metals, indium, nickel, cobalt, arsenic, antimony, lead, and zinc.

A simplified flow diagram identifying solid waste sources and disposal is shown in Figure 1. The primary process steps comprise (1) roasting to reduce sulfur content, (2) reverberatory furnace smelting to form copper sulfide matte and a siliceous slag which is discarded, (3) oxidation (blowing) the molten sulfide matte to form molten "blister" copper and an iron silicate slag which is returned to the reverberatory, and (4) furnace purification (fire-refining) of the molten copper such that anodes suitable for electrolytic refining can be cast (or copper product can be marketed directly).

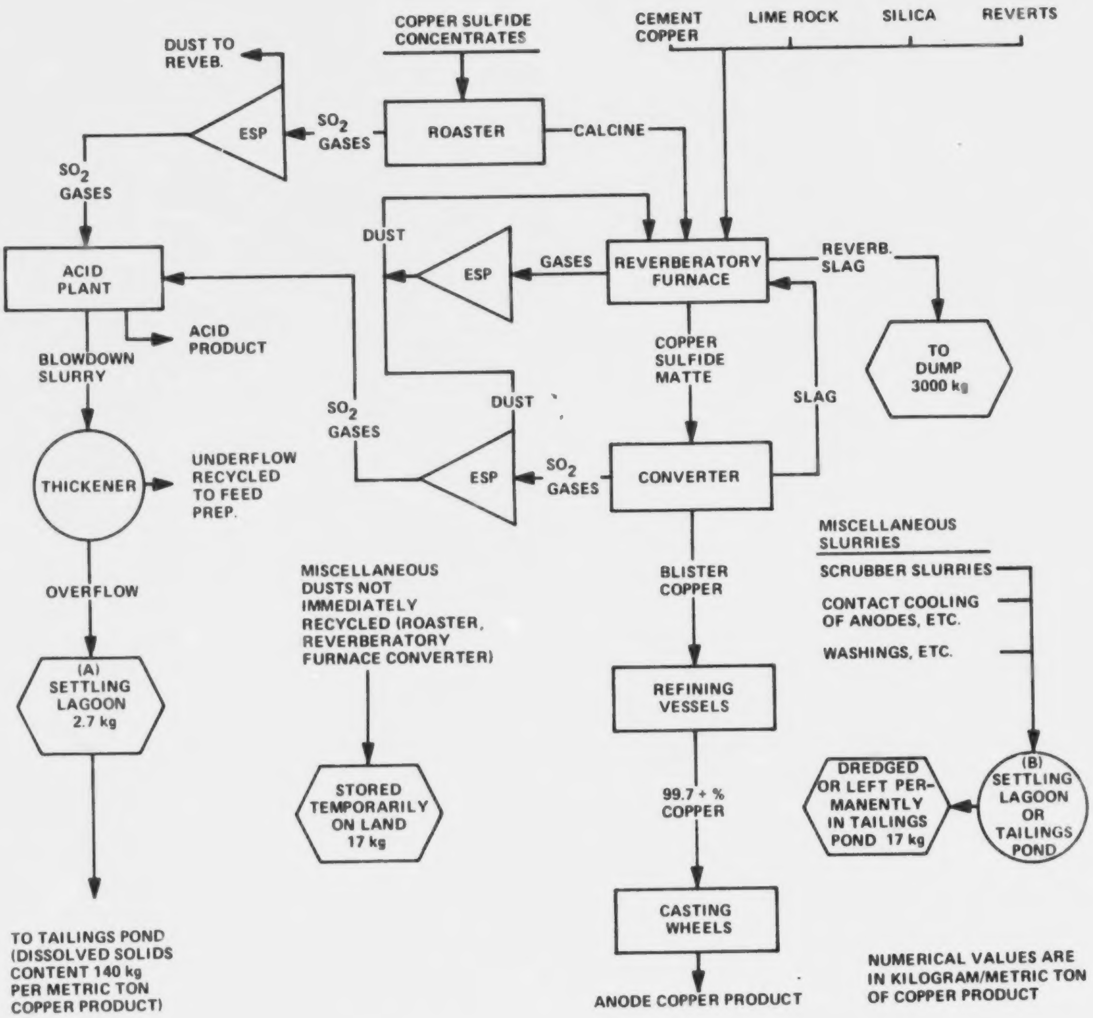
Roaster gases are rich in SO_2 such that sulfuric acid can be produced in large quantities; some of the acid is used in the adjacent electrolytic refinery. Some copper smelters do not roast prior to reverberatory furnace smelting but do make sulfuric acid from converter gases; some smelters do not have adjacent electrolytic copper refineries. Acid plant blowdown slurry effluent yields a sludge which is not 100 percent recycled, as will be discussed in a following section.

Electrolytic Refining. The hypothetical copper electrolytic refining plant is assumed to have an annual capacity of 160,000 MT (176,000 short tons) per year. This is the approximate average of all electrolytic refinery capacities in the U.S. Copper refineries are estimated to operate 350 days per year; the daily capacity is, therefore, 457 MT (503 short tons) per day.

Electrolytic refinery feed material consists solely of cast copper anodes having a purity in the range 99.0 to 99.7 percent copper. The cathode copper product resulting from electrolysis has a purity of the order of 99.95 percent.

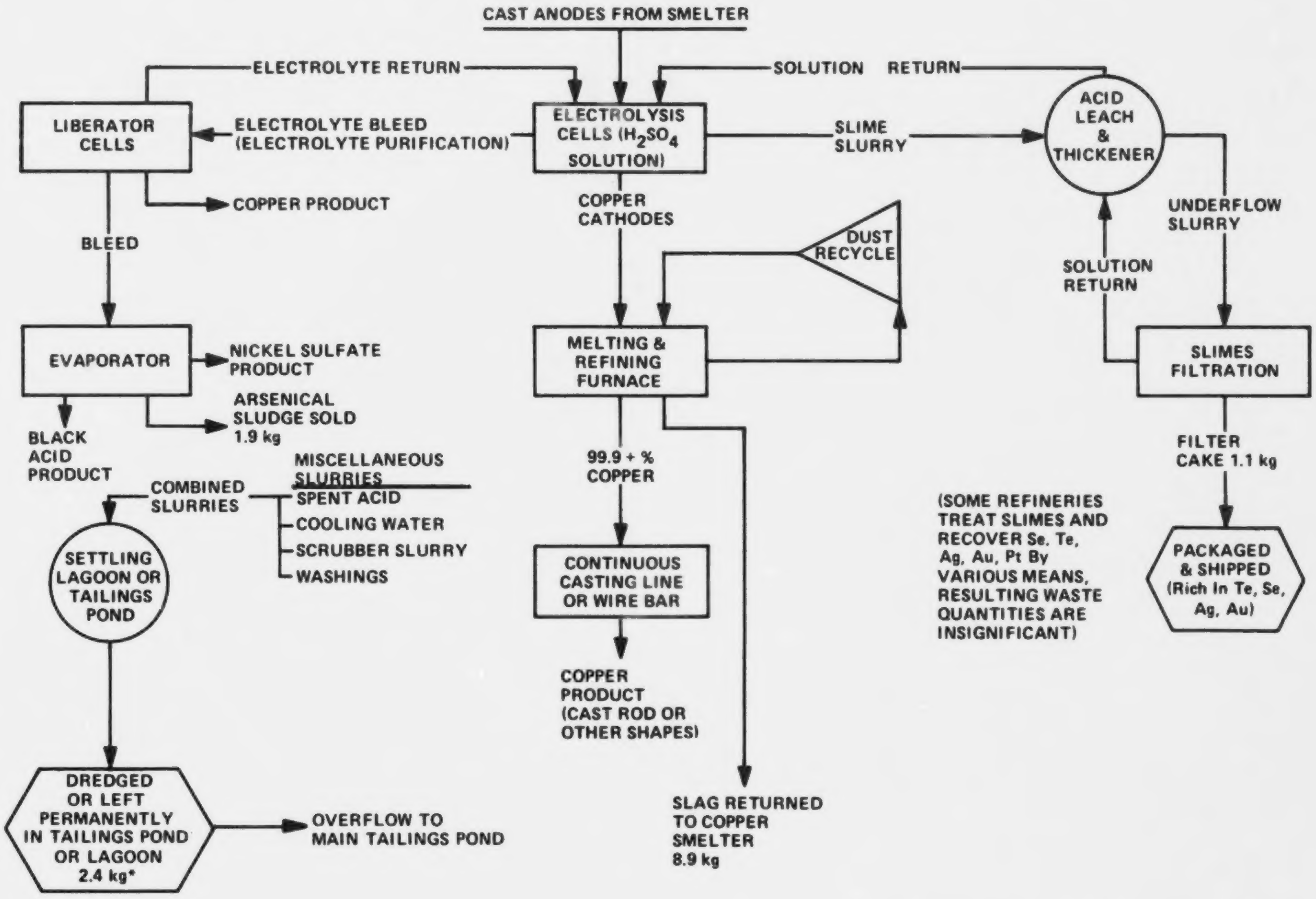
A simplified flow diagram identifying solid waste sources and disposal is shown in Figure 2. The primary process steps comprise (1) electrolysis in large lead or plastic lined cells, (2) melting and refining with respect to sulfur, hydrogen, and oxygen, and (3) casting into shapes such as wirebar, or into continuous cast rod for the wire drawing industry.

Important auxiliary process steps are slimes recovery from the cell bottoms, and electrolyte purification to permit electrolyte reuse and to recover materials of value. The slimes tend to be rich in Se, Te, As, Ag, Au and Pt such that their value is very high; the typical refinery ships the slimes filter cakes elsewhere for treatment to recovery these metals. (Three of the existing electrolytic copper refineries in the U.S. treat slimes from their own operations and from other refineries). Since slimes yield is only of the order of 0.001 metric ton per metric ton of copper refined, it is apparent that the slimes recovery "industry" in the U.S. is very small relative to copper production. Considering this and the fact that the only solid residue


COPPER SULFIDE CONCENTRATES
CEMENT COPPER
LIME ROCK
SILICA
REVERTS
DUST TO REVEB.
ESP
SO_2 GASES
ROASTER
CALCINE
SO_2 GASES
DUST
ACID PLANT
ESP
GASES
REVERBERATORY FURNACE
REVERB. SLAG
ACID PRODUCT
BLOWDOWN SLURRY
COPPER SULFIDE MATTE
TO DUMP 3000 kg
SO_2 GASES
DUST
SLAG
THICKENER
UNDERFLOW RECYCLED TO FEED PREP.
ESP
SO_2 GASES
CONVERTER
MISCELLANEOUS SLURRIES
OVERFLOW
MISCELLANEOUS DUSTS NOT IMMEDIATELY RECYCLED (ROASTER, REVERBERATORY FURNACE CONVERTER)
BLISTER COPPER
SCRUBBER SLURRIES
CONTACT COOLING OF ANODES, ETC.
WASHINGS, ETC.
(A) SETTLING LAGOON 2.7 kg
REFINING VESSELS
STORED TEMPORARILY ON LAND 17 kg
99.7 + % COPPER
DREDGED OR LEFT PERMANENTLY IN TAILINGS POND 17 kg
(B) SETTLING LAGOON OR TAILINGS POND
CASTING WHEELS
TO TAILINGS POND (DISSOLVED SOLIDS CONTENT 140 kg PER METRIC TON COPPER PRODUCT)
ANODE COPPER PRODUCT

NUMERICAL VALUES ARE IN KILOGRAM/METRIC TON OF COPPER PRODUCT

Figure 1 PRIMARY COPPER SMELTING AND FIRE REFINING


CAST ANODES FROM SMELTER
ELECTROLYTE RETURN
SOLUTION RETURN
LIBERATOR CELLS
ELECTROLYTE BLEED (ELECTROLYTE PURIFICATION)
ELECTROLYSIS CELLS (H_2SO_4 SOLUTION)
SLIME SLURRY
ACID LEACH & THICKENER
COPPER PRODUCT
BLEED
COPPER CATHODES
DUST RECYCLE
UNDERFLOW SLURRY
SOLUTION RETURN
EVAPORATOR
NICKEL SULFATE PRODUCT
MELTING & REFINING FURNACE
SLIMES FILTRATION
ARSENICAL SLUDGE SOLD 1.9 kg
BLACK ACID PRODUCT
99.9 + % COPPER
FILTER CAKE 1.1 kg
MISCELLANEOUS SLURRIES
COMBINED SLURRIES
SPENT ACID
COOLING WATER
SCRUBBER SLURRY
WASHINGS
(SOME REFINERIES TREAT SLIMES AND RECOVER Se. Te, Ag, Au, Pt By VARIOUS MEANS, RESULTING WASTE QUANTITIES ARE INSIGNIFICANT)
SETTLING LAGOON OR TAILINGS POND
CONTINUOUS CASTING LINE OR WIRE BAR
PACKAGED & SHIPPED (Rich In Te, Se, Ag, Au)
COPPER PRODUCT (CAST ROD OR OTHER SHAPES)
DREDGED OR LEFT PERMANENTLY IN TAILINGS POND OR LAGOON 2.4 kg*
OVERFLOW TO MAIN TAILINGS POND
SLAG RETURNED TO COPPER SMELTER 8.9 kg

*TOTAL OF SUSPENDED & DISSOLVED SOLIDS IN EFFLUENT SLURRY

NUMERICAL VALUES ARE IN KILOGRAMS/METRIC TON OF COPPER PRODUCT

Figure 2 PRIMARY ELECTROLYTIC COPPER REFINING

of note produced in slimes treatment is a slag which is recycled to a copper smelter, it is clear that the total amount of solid waste generated in slimes treatment is insignificant in the context of the present study.

Electrolyte purification consists of the following steps performed on the bleed stream of impure sulfuric acid containing copper and other elements in solution: (1) copper removal by electrolysis with insoluble lead anodes in liberator cells, (2) filtration to remove an arsenical sludge, and (3) evaporation to precipitate nickel sulfate. A "black acid" product remains after nickel sulfate removal. This is marketed for acid recovery.

Slags taken from the melting and refining furnace (and from the anode casting furnace at refineries which cast anodes) are very rich in copper (50% Cu), i.e., immediate recycle to a copper smelter is dictated and practiced.

1.2.2 Description of Waste Streams

Smelting and Fire Refining. Three general types of solid residues containing potentially hazardous elements are found at copper smelters, namely, slag, dry dusts, and wet sludges and slurries. The amount of slag produced and disposed per unit of copper production can be accurately estimated from the data obtained from nine smelters by Calspan. Average factors for the estimating generation or disposal of dusts and sludges may be departed from significantly at individual plants for the following reasons: (1) the amount of dust generated and collected varies greatly with smelting practice and dust collection practice; (2) complete recycle of dusts may prevail at many or all smelters, but there are at least some instances in which dusts are stored on the ground prior to recycle; (3) sludges stemming from slurries generated by operations, such as acid plant blowdown, contact cooling and washdown, appear to vary widely in amount and disposition. For example, the more effective the electrostatic precipitator preceding the acid plant, the less will be the particulate content of the blowdown from the acid plant, and probably the less will be the volume of the blowdown stream necessary to maintain the acid plant in operating equilibrium. However, even with less effective electrostatic precipitators, if the thickener on the blowdown stream is effective and if the particulate-laden underflow from the thickener is recycled directly to, say, reverberatory feed preparation, the amount of suspended and dissolved solids going to a settling pond, neutralization plant, and/or tailings pond may be relatively small; (4) acid effluent generation and neutralization practice is currently in a state of very rapid change at a high percentage of the copper smelters of the United States. This situation arises mainly from the enforcement of air and water pollution regulation. Many acid plants have been added to reduce SO_2 emission from stacks, and lime neutralization plants are being added to reduce the acid and dissolved metal contents of effluents to streams and to facilitate total recycle of water.

In view of the above, the following solid residue generation and disposal factors are presented as shown in Figure 1.

a. Reverberatory Slag

The chief solid waste from the hypothetical copper smelter is reverberatory furnace slag amounting to 3000 kg/MT of refined copper or 295,000 MT/year. Molten slag is transported about 600 m (2000 ft) to the main dump by refractory-lined rail-car ladles. The accumulated slag pile has a volume of the order of 1.5 x 10^6 cu m (2 x 10^6 cu yards).

b. Dusts From Roasters, Reverbatory Furnaces and Converters

Large amounts of dust are collected from each of the primary process operations and are generally recycled without temporary storage on land. It is estimated, however, that 17 kg of dust/MT of refined copper or 4.8 MT/day are stored on land near the smelter prior to eventual recycle. The size of the storage pile is estimated to be approximately 150 MT.

c. Acid Plant Blowdown Slurry (Sludge)

Blowdown from the acid plant amounts to 7,940 liter/MT of copper product or 2,270 cu m/day (600,000 gal/day). The blowdown stream flows to a thickener which yields an underflow of 82 cu m/day (21,600 gal/day) containing 46 MT/day of suspended solids. All underflow is recycled immediately to feed preparation, it is in no sense a waste.

Overflow from the thickener amounts to 2,200 cu m/day (570,000 gal/day), containing 0.77 MT/day suspended solids and 40 MT/day dissolved solids. The suspended solids are settled in an unlined lagoon, dredged periodically, stored on the ground near the lagoon, and eventually recycled.

Effluent from the lagoon goes to the main tailings pond associated with the mine-mill-smelter complex. This effluent contains 140 kg of dissolved solids per MT copper product or 40 MT/day dissolved solids which tend to be precipitated to an unknown degree in the tailings pond, by reaction with the alkaline tailings and/or evaporation. The weight or volume of tailings is overwhelmingly greater than that of the precipitated solids, but the dissolved solids from acid plant blowdown are nonetheless present. (No significant portion of the cost of the tailings pond is attributable to smelter waste disposal because of overwhelming volume of tailings waste).

d. Miscellaneous Slurries (scrubbers, contact cooling, washing)

Effluents arising from sources such as wet scrubbers, anode cooling, other contact cooling water, and plant washdown carry dissolved and suspended solids amounting to 4.8 MT/day. The dredged sludge amounting to 17 kg/MT of copper is dried on the ground near the lagoon and eventually recycled. It is estimated that the average size of the accumulation drying on the ground at any time is 800 MT. The dissolved solid content of this effluent stream is unknown; the lagoon overflow goes to the main tailings pond where some degree of precipitation of dissolved solids would be expected as discussed above.

Electrolytic Refining.

(a) Miscellaneous Slurries. The only waste ultimately land disposed or stored on land in electrolytic copper refining is a combined dilute slurry composed of effluents from operations such as contact cooling of cast copper anodes and/or product, non-contact cooling of furnaces, spent anode washing, and plant washdown. This dilute acid slurry containing suspended and dissolved solids goes to an unlined settling lagoon approximately one-half acre in area. Flow is 6,800 cu m/day, suspended solids content is 210 kg/day, and dissolved solids content is 870 kg/day. The suspended solids settle in the lagoon and are dredged as necessary and disposed of on land. Dissolved solids pass on with the overflow to the main tailings pond (associated with the nearby ore concentrator) where an unknown degree of precipitation would be expected due to evaporation and reaction with the tailings. At some refinereies located on navigable waters and not on-site with ore concentrators, the dilute slurry effluent goes directly to a lagoon of roughly four acre-ft capacity adjacent to the bay or river. An unknown degree of settling and precipitation would occur in such a lagoon. While this practice is presently in existence, it is believed that existing or soon-to-exist effluent control regulations require lime neutralization and impoundment of the resulting sludge in such cases.

In the case of discharge to the concentrator tailings pond, the solids buildup attributable to the above-described dilute slurry is insignificant relative to that due to ore tailings. These solids, however, contain potentially hazardous heavy metals.

(b) Other Residues. As indicated in Figure 2, all other significant residues associated with electrolytic copper refining are immediately recycled or shipped elsewhere for recovery of metal values. The arsenical sludge resulting from electrolyte purification is stored in concrete bins and shipped elsewhere for processing. Slimes recovered from the electrolytic cells are of extremely high value, and the slag from the melting furnaces is shipped to a copper smelter for reprocessing.

Analyses of samples collected from copper smelters and refineries are given in Appendix A. Not all copper smelting and refining wastes are considered as having equal degrees of potential hazard. Copper smelter slags are large chunks of hard siliceous furnace residue. In solubility tests described in Appendix B it was found that these slags did not leach significantly. Therefore, copper slags (reverbatory and electric furnace) are considered non-hazardous in view of data available at this time.

In solubility tests it was found that sludges (acid plant sludge) and dusts (converter dust, reverbatory dust) leached copper, lead, zinc, and cadmium in significant concentrations (see Appendix B). Leaching of dusts and sludges would be anticipated in view of the fine particle size (i.e., large exposed surface area). These wastes are therefore all considered potentially hazardous.

1.2.3 Waste Quantities

Table 3 summarizes the generation factors for each of the land disposed or stored residuals in the primary copper smelting and fire refining category. This table also gives typical concentrations of potentially hazardous constituents. Table 4 gives the yearly quantity of wastes and potentially hazardous constituents for a typical plant producing 100,000 MT (110,000 short tons) metric tons of blister copper (i.e. from smelter) and 100,000 MT (110,000 short tons) tons of fire refined copper per year.

Table 5 summarizes the generation factors for each of the land disposed or stored residuals in the primary copper electrolytic refining category. Table 6 gives the yearly quantity of wastes and potentially hazardous constituents for a typical plant producing 160,000 MT (176,000 short tons) of electrolytic copper per year.

Using the waste generation factors and concentration factors contained in Tables 3 and 5 and copper smelter and refinery production capacities, estimates have been made for the quantities of land disposed or stored waste and hazardous constituents thereof. These estimates are given for 1974, 1977, and 1983 on a state-by-state, regional and national level in Tables 7 through 8. It is seen in these tables that reverbatory slag is generated in huge amounts compared to sludges and dusts. However, as previously discussed, this waste is not considered potentially hazardous.

Sludges from smelters and fire refiners are generated at a much higher rate than those from electrolytic refiners mainly as a result of the need to scrub SO_2 from emissions of smelters. Electrolytic refiners produce relatively little sludge. Increases in the quantities of sludges from milling and fire refining and electrolytic refining will increase moderately over the period 1974 to 1983 paralleling predicted increases in production capacity.

TABLE 3 WASTE RESIDUAL FACTORS, COPPER SMELTERS AND FIRE REFINERS

Type of Waste	Generation Factor Kg/MT	Concentration Factor (ppm) Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
Slag	3000	10	110	7,430	0.8	280	18	143	250	27	4,040
Sludges	155	520	50	279,400	0.8	89	110	8,000	500	30	27,900
Dusts	17	520	88	37,260	13	260	90	1,320	143	33	2,250

TABLE 4 YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL PLANT, COPPER SMELTERS AND FIRE REFINERS

Type of Waste	Total Waste Quantity (MT)	Quantity of Potentially Hazardous Constituents (MT) Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
Slag	300,000	3.0	3300	2230	0.24	84	5.4	43	75	8.1	1210
Sludges	15,500	8.1	0.78	4330	0.012	1.4	1.7	124	7.8	0.47	432
Dusts	1,700	0.88	0.15	63	0.022	0.44	0.15	2.2	0.24	0.056	3.8

Quantities calculated from generation and concentration factors given in Table 3 based on annual production of 100,000 MT. Divide by 350 to obtain daily quantities, multiply by 1.1 to conver to short tons.

TABLE 5 WASTE RESIDUAL FACTORS, COPPER ELECTROLYTIC REFINERIES

Type of Waste	Generation Factor Kg/MT	Concentration Factor (ppm)									
		Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
Sludge	2.4	180	25.0	22,000	5.0	8.0	10.0	12,000	800	550	190

TABLE 6 YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL PLANT, COPPER ELECTROLYTIC REFINERIES

Type of Waste	Total Waste Quantity (MT)	Quantity of Potentially Hazardous Constituents (MT)									
		Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
Sludge	384	0.069	0.0096	8.4	0.0019	0.0031	0.0038	4.6	0.31	0.21	0.073

Quantities calculated from generation and concentration factors given in Table
based on annual production of 160,000 MT
Divide by 350 to obtain daily quantities, multiply by 1.1 to convert to short tons.

Table 7-a
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY COPPER SMELTERS AND FIRE REFINERS TOTAL SLAG - 1974 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	DISPOSAL METHOD	CONSTITUENTS									
				Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	482,600	0	ALL DISPOSED IN OPEN DUMPS	4.8	52	3600	0.39	135	8.8	69	121	12.9	1950
TENNESEE	44,300			0.4	5	300	0.04	12	0.8	6	11	1.2	180
MICHIGAN	314,000			3.1	34	2300	0.24	88	5.7	45	79	8.4	1270
TEXAS	233,100			2.3	25	1700	0.19	65	4.3	33	58	6.2	940
NEW MEXICO	236,600			2.7	25	1800	0.19	66	4.3	34	59	6.3	960
MONTANA	541,700			5.4	58	4000	0.43	152	9.9	78	135	14.5	2190
UTAH	580,000			5.8	62	4300	0.46	162	10.6	83	145	15.5	2350
ARIZONA	3,022,700			30.2	324	22500	2.42	847	55.3	433	756	80.8	12200
NEVADA	115,300			1.2	12	900	0.09	32	2.1	17	29	3.1	470
WASHINGTON	177,300			1.8	19	1300	0.14	50	3.2	25	44	4.7	720
EPA REGION II	483,000			4.8	52	3600	0.39	135	8.8	69	121	12.9	1950
IV	44,000			0.4	5	300	0.04	12	0.8	6	11	1.2	180
V	314,000			3.1	34	2300	0.24	88	5.7	45	79	8.4	1270
VI	470,000			5.0	50	3500	0.38	131	8.6	67	117	12.5	1900
VIII	1,122,000			11.2	120	8300	0.89	314	20.5	161	280	30.0	4540
IX	3,138,000			31.4	336	23400	2.51	879	57.4	450	785	83.9	12670
X	177,000			1.8	19	1300	0.14	50	3.2	25	44	4.7	720
NATIONAL TOTAL	5,748,000			57.7	616	42700	4.59	1609	105	823	1437	153.6	23230

*THIS WASTE BELIEVED NONHAZARDOUS BASED ON CALSPAN SOLUBILITY TESTS DESCRIBED IN APPENDIX B.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 7-b
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY COPPER SMELTERS AND FIRE REFINERS TOTAL SLAG - 1977, 1983 (METRIC TONS)*

STATE	TOTAL DISPOSED (A)	TOTAL POTENTIALLY HAZARDOUS	DISPOSAL METHOD	CONSTITUENTS									
				Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	482,600	0	ALL DISPOSED IN OPEN DUMPS	4.8	52	3,600	0.39	135	8.8	69	121	12.9	1,950
TENNESEE	44,300			0.4	5	300	0.04	12	0.8	6	11	1.2	180
MICHIGAN	314,000			3.1	34	2,300	0.24	88	5.7	45	79	8.4	1,270
TEXAS	233,100			2.3	25	1,700	0.19	65	4.3	33	58	6.2	940
NEW MEXICO	473,200			5.4	50	3,600	0.38	132	8.6	68	118	12.6	1,920
MONTANA	541,700			5.4	58	4,000	0.43	152	9.9	78	135	14.5	2,190
UTAH	580,000			5.8	62	4,300	0.46	162	10.6	83	145	15.5	2,350
ARIZONA	3,022,700			30.2	324	22,500	2.42	847	55.3	433	756	80.8	12,200
NEVADA	115,300			1.2	12	900	0.09	32	2.1	17	29	3.1	470
WASHINGTON	177,300			1.8	19	1,300	0.14	50	3.2	25	44	4.7	720
EPA REGION II	483,000			4.8	52	3,600	0.39	135	8.8	69	121	12.9	1,950
IV	44,000			0.4	5	300	0.04	12	0.8	6	11	1.2	180
V	314,000			3.1	34	2,300	0.24	88	5.7	45	79	8.4	1,270
VI	706,600			7.7	75	5,300	0.57	197	12.9	101	176	18.8	2,860
VIII	1,122,000			11.2	120	8,300	0.89	314	20.5	161	280	30.0	4,540
IX	3,138,000			31.4	336	23,400	2.51	879	57.4	450	785	83.9	12,670
X	177,000			1.8	19	1,300	0.14	50	3.2	25	44	4.7	720
NATIONAL TOTAL	5,984,600			60.4	641	44,500	4.78	1,675	109.3	857	1,496	159.9	24,190

*THIS WASTE BELIEVED NONHAZARDOUS BASED ON CALSPAN SOLUBILITY TESTS DESCRIBED IN APPENDIX B.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 7-c
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTES FROM PRIMARY COPPER SMELTERS AND FIRE REFINERS TOTAL SLUDGES - 1974 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS									
					Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	25,400	25,400	1,071.14	ALL TO LAGOONS (A) ↓	13.2	2.2	950	0.31	6.7	2.3	35	3.6	0.83	57
TENNESSEE	2,300	2,300	100.61		1.2	0.2	90	0.03	0.6	0.2	3	0.3	0.08	5
MICHIGAN	17,500	17,500	733.10		9.1	1.5	650	0.22	4.6	1.6	24	2.5	0.58	39
TEXAS	14,800	14,800	620.97		7.7	1.3	550	0.18	3.9	1.3	21	2.1	0.49	33
NEW MEXICO	12,400	12,400	519.36		6.5	1.1	460	0.15	3.3	1.1	17	1.8	0.41	28
MONTANA	28,500	28,500	1,186.89		14.9	2.5	1,060	0.35	7.5	2.6	40	4.1	0.94	54
UTAH	34,300	34,300	1,444.56		17.9	3.0	1,280	0.43	9.1	3.1	48	4.9	1.13	77
ARIZONA	143,600	143,600	6,028.43		74.9	12.7	5,350	1.80	37.9	12.9	190	20.5	4.73	323
NEVADA	6,100	6,100	257.98		3.2	0.5	230	0.08	1.6	0.5	8	0.9	0.20	13
WASHINGTON	14,100	14,100	598.23		7.4	1.2	530	0.17	3.7	1.3	20	2.0	0.46	32
EPA REGION II	25,400	25,400	1,071.14		13.2	2.2	950	0.31	6.7	2.3	35	3.6	0.83	57
IV	2,300	2,300	100.61		1.2	0.2	90	0.03	0.6	0.2	3	0.3	0.08	5
V	17,500	17,500	733.10		9.1	1.5	650	0.22	4.6	1.6	24	2.5	0.58	39
VI	27,200	27,200	1,140.33		14.2	2.4	1,010	0.33	7.2	2.4	38	3.9	0.90	61
VIII	62,700	62,700	2,631.45		32.8	5.5	2,340	0.78	16.6	5.7	88	9.0	2.07	131
IX	149,700	149,700	6,286.41		78.1	13.2	5,580	1.88	39.5	13.4	198	21.4	4.93	336
X	14,100	14,100	598.23		7.4	1.2	530	0.17	3.7	1.3	20	2.0	0.46	32
NATIONAL TOTAL	298,900	298,900	12,561.27		156	26.2	11,150	3.72	78.9	26.9	406	42.7	9.85	661

(A) A PORTION OF THESE SLUDGES INCLUDING ACID PLANT SLUDGE IS EVENTUALLY RECYCLED AFTER LAND DRYING AND STORAGE; THE REMAINDER IS PERMANENTLY DISPOSED IN LARGE TAILINGS PONDS OR THE LIKE.

*MULTIPLY BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 7-d
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTES FROM PRIMARY COPPER SMELTERS AND FIRE REFINERS TOTAL SLUDGES - 1977 (METRIC TONS, DRY WEIGHT)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS									
					Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	25,400	25,400	1,071	ALL TO	13.2	2.2	950	0.31	6.7	2.3	35	3.6	0.83	57
TENNESSEE	2,300	2,300	101	LAGOONS	1.2	0.2	90	0.03	0.6	0.2	3	0.3	0.08	5
MICHIGAN	17,500	17,500	733	(A)	9.1	1.5	650	0.22	4.6	1.6	24	2.5	0.58	39
TEXAS	14,800	14,800	621	↓	7.7	1.3	550	0.18	3.9	1.3	21	2.1	0.49	33
NEW MEXICO	24,800	24,800	1,039		13.0	2.2	920	0.30	6.6	2.2	34	3.6	0.82	56
MONTANA	28,500	28,500	1,187		14.9	2.5	1,060	0.35	7.5	2.6	40	4.1	0.94	54
UTAH	34,300	34,300	1,445		17.9	3.0	1,280	0.43	9.1	3.1	48	4.9	1.13	77
ARIZONA	143,600	143,600	6,028		74.9	12.7	5,350	1.80	37.9	12.9	190	20.5	4.73	323
NEVADA	6,100	6,100	258		3.2	0.5	230	0.08	1.6	0.5	8	0.9	0.20	13
WASHINGTON	14,100	14,100	598		7.4	1.2	530	0.17	3.7	1.3	20	2.0	0.46	32
EPA REGION II	25,400	25,400	1,071		13.2	2.2	950	0.31	6.7	2.3	35	3.6	0.83	57
IV	2,300	2,300	101		1.2	0.2	90	0.03	0.6	0.2	3	0.3	0.08	5
V	17,500	17,500	733		9.1	1.5	650	0.22	4.6	1.6	24	2.5	0.58	39
VI	39,600	39,600	1,660		20.7	3.5	1,470	0.48	10.5	3.5	55	5.7	1.31	89
VIII	62,700	62,700	2,631		32.8	5.5	2,340	0.78	16.6	5.7	88	9.0	2.07	131
IX	149,700	149,700	6,286		78.1	13.2	5,580	1.88	39.5	13.4	198	21.4	4.93	336
X	14,100	14,100	598		7.4	1.2	530	0.17	3.7	1.3	20	2.0	0.46	32
NATIONAL TOTAL	311,300	311,300	13,081		162.5	27.3	11,610	3.87	82.2	28.0	423	44.5	10.26	689

(A) A PORTION OF THESE SLUDGES INCLUDING ACID PLANT SLUDGE IS EVENTUALLY RECYCLED AFTER LAND DRYING AND STORAGE. THE REMAINDER IS PERMANENTLY DISPOSED IN LARGE TAILINGS PONDS OR THE LIKE.

*MULTIPLY BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 7-e
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTES FROM PRIMARY COPPER SMELTERS AND FIRE REFINERS TOTAL SLUDGES - 1983 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS									
					Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	32,004	32,004	1,350	ALL TO LAGOONS (A) ↓	16.6	2.8	1,197	0.39	8.4	2.9	44	4.5	1.05	72
TENNESSEE	2,898	2,898	127		1.5	0.3	113	0.04	0.8	0.3	0.4	0.4	0.10	6
MICHIGAN	22,050	22,050	924		11.5	1.9	819	0.28	5.8	2.0	30	3.2	0.73	49
TEXAS	18,648	18,648	782		9.7	1.6	693	0.23	4.9	1.6	26	2.6	0.62	42
NEW MEXICO	31,248	31,248	1,309		16.4	2.8	1,159	0.38	8.3	2.8	43	4.5	1.03	71
MONTANA	35,910	35,910	1,495		18.8	3.2	1,336	0.44	9.5	3.3	50	5.2	1.18	68
UTAH	43,218	43,218	1,820		22.6	3.8	1,613	0.54	11.5	3.9	60	6.2	1.42	97
ARIZONA	180,936	180,936	7,596		94.4	16.0	6,741	2.27	47.8	16.3	239	25.8	5.96	407
NEVADA	7,686	7,686	325		4.0	0.6	290	0.10	2.0	0.6	10	1.1	0.25	16
WASHINGTON	17,766	17,766	754		9.3	1.5	668	0.21	4.7	1.6	25	2.5	0.58	40
EPA REGION II	32,004	32,004	1,350		16.6	2.8	1,197	0.39	8.4	2.9	44	4.5	1.05	72
IV	2,898	2,898	127		1.5	0.3	113	0.04	0.8	0.3	0.4	0.4	0.10	6
V	22,050	22,050	924		11.5	1.9	819	0.28	5.8	2.0	30	3.2	0.73	49
VI	49,896	49,896	2,091		26.1	4.4	1,852	0.61	13.2	4.4	69	7.1	1.65	113
VIII	79,128	79,128	3,316		41.4	7.0	2,949	0.98	21.0	7.2	110	11.4	2.60	165
IX	188,622	188,622	792		98.4	16.6	7,031	2.37	49.8	16.9	249	26.9	6.21	423
X	17,766	17,766	754		9.3	1.5	668	0.21	4.7	1.6	25	2.5	0.58	48
NATIONAL TOTAL	392,364	392,364	16,482		204.8	34.5	14,629	4.88	103.7	35.3	531	56.0	12.72	868

(A) A PORTION OF THESE SLUDGES INCLUDING ACID PLANT SLUDGE IS EVENTUALLY RECYCLED AFTER LAND DRYING AND STORAGE; THE REMAINDER IS PERMANENTLY DISPOSED IN LARGE TAILINGS PONDS OR THE LIKE.

*MULTIPLY BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 7-f

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY COPPER SMELTERS AND FIRE REFINERS TOTAL DUSTS - 1974 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS										
					As	Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	2780	2780	883.545	LAND STORAGE (A) ↓		1.4	0.14	780	0.0020	0.22	0.30	22	1.4	0.083	78
TENNESSEE	750	750	238.0126			0.4	0.04	210	0.0006	0.07	0.08	6	0.4	0.022	21
MICHIGAN	5340	5340	1,699.0043			2.8	0.27	1500	0.0043	0.48	0.59	43	2.7	0.160	149
TEXAS	1620	1620	510.0603			0.8	0.08	450	0.0013	0.15	0.18	13	0.8	0.049	45
NEW MEXICO	1360	1360	430.7721			0.7	0.07	380	0.0011	0.12	0.14	11	0.7	0.041	38
MONTANA	3120	3120	985.0765			1.6	0.16	870	0.0025	0.28	0.34	24	1.6	0.094	87
UTAH	3760	3760	1,190.956			2.0	0.19	1050	0.0030	0.34	0.41	31	1.9	0.113	105
ARIZONA	15,700	15,700	4,986.5136			8.2	0.79	4400	0.0126	1.41	1.73	126	7.9	0.471	440
NEVADA	660	660	203.7795			0.3	0.03	180	0.0005	0.06	0.07	5	0.3	0.019	18
WASHINGTON	1550	1550	487.0382			0.8	0.08	430	0.0012	0.14	0.17	12	0.8	0.047	43
EPA REGION II	2780	2780	883.545			1.4	0.14	780	0.0020	0.22	0.30	22	1.4	0.083	78
IV	750	750	238.0126			0.4	0.04	210	0.0006	0.07	0.08	6	0.4	0.022	21
V	5340	5340	1,699.0043			2.8	0.27	1500	0.0043	0.48	0.59	43	2.7	0.160	149
VI	2980	2980	940.8324			1.5	0.15	830	0.0024	0.27	0.32	24	1.5	0.090	83
VIII	6880	6880	2,176.0325			3.6	0.35	1920	0.0055	0.62	0.75	55	3.5	0.207	192
IX	16,400	16,400	5,190.2931			8.5	0.82	4580	0.0131	1.47	1.80	131	8.2	0.490	458
X	1550	1550	487.0382			0.8	0.08	430	0.0012	0.14	0.17	12	0.8	0.047	43
NATIONAL TOTAL	36,680	36,630	11,614.7581			19.0	1.85	10250	0.0291	3.27	4.01	293	18.5	1.099	1024

(A) THESE DUSTS ARE DEFINITELY RECYCLED BUT ARE STORED ON LAND TEMPORARILY FOR A NUMBER OF MONTHS.

* MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

Table 7-g
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY COPPER SMELTERS AND FIRE REFINERS TOTAL DUSTS - 1977, 1983 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS										
					As	Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	2,780	2,780	883	LAND STORAGE (A)		1.4	0.14	780	0.0020	0.22	0.30	22	1.4	0.083	78
TENNESSEE	750	750	238			0.4	0.04	210	0.0006	0.07	0.08	6	0.4	0.022	21
MICHIGAN	5,340	5,340	1,699			2.8	0.27	1,500	0.0043	0.48	0.59	43	2.7	0.160	149
TEXAS	1,620	1,620	510			0.8	0.08	450	0.0013	0.15	0.18	13	0.8	0.049	45
NEW MEXICO	2,720	2,720	861			1.4	0.14	760	0.0022	0.24	0.28	22	1.4	0.082	76
MONTANA	3,120	3,120	985			1.6	0.16	870	0.0025	0.28	0.34	24	1.6	0.094	87
UTAH	3,760	3,760	1,191			2.0	0.19	1,050	0.0030	0.34	0.41	31	1.9	0.113	105
ARIZONA	15,700	15,700	4,986			8.2	0.79	4,400	0.0126	1.41	1.73	126	7.9	0.471	440
NEVADA	660	660	204			0.3	0.03	180	0.0005	0.06	0.07	5	0.3	0.019	18
WASHINGTON	1,550	1,550	487			0.8	0.08	430	0.0012	0.14	0.17	12	0.8	0.047	43
EPA REGION II	2,780	2,780	883			1.4	0.14	780	0.0020	0.22	0.30	22	1.4	0.083	78
IV	750	750	238			0.4	0.04	210	0.0006	0.07	0.08	6	0.4	0.022	21
V	5,340	5,340	1,699			2.8	0.27	1,500	0.0043	0.48	0.59	43	2.7	0.160	149
VI	4,340	4,340	1,372			2.1	0.22	1,210	0.0035	0.39	0.46	35	2.7	0.131	121
VII	6,880	6,880	2,176			3.6	0.35	1,920	0.0055	0.62	0.75	55	3.5	0.207	192
IX	16,400	16,400	5,190			8.5	0.82	4,580	0.0131	1.47	1.80	131	8.2	0.490	458
X	1,550	1,550	487			0.8	0.08	430	0.0012	0.14	0.17	12	0.8	0.047	43
NATIONAL TOTAL	38,040	38,040	12,045			19.7	1.92	10,630	0.0401	3.39	4.15	304	19.2	1.140	1,062

(A) THESE DUSTS ARE DEFINITELY RECYCLED BUT ARE STORED ON LAND TEMPORARILY FOR A NUMBER OF MONTHS.

* MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

Table 8-a
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY COPPER ELECTROLYTIC REFINERS
TOTAL SLUDGE - 1974 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENT (B)									
					Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	1,180	1,180	44.2571	ALL TO LAGOONS (A) ↓	0.21	0.030	26	0.0059	0.0094	0.0118	14.2	0.94	0.65	2.2
NEW YORK	170	170	6.5679		0.03	0.004	4	0.0008	0.0014	0.0017	2.0	0.14	0.09	0.3
MARYLAND	1,250	1,250	47.3798		0.23	0.031	28	0.0063	0.0100	0.0125	15.0	1.00	0.69	2.4
TEXAS	1,000	1,000	37.478		0.18	0.025	22	0.0050	0.0080	0.0100	12.0	0.80	0.55	1.9
MISSOURI	110	110	3.6755		0.02	0.003	2	0.0005	0.0009	0.0011	1.3	0.09	0.06	0.2
MONTANA	460	460	17.1226		0.08	0.012	10	0.0023	0.0037	0.0046	5.5	0.37	0.25	0.9
UTAH	470	470	17.2429		0.08	0.012	10	0.0024	0.0038	0.0047	5.6	0.38	0.26	0.9
ARIZONA	650	650	24.0310		0.12	0.016	14	0.0033	0.0052	0.0065	7.8	0.52	0.36	1.2
WASHINGTON	300	300	11.6749		0.05	0.008	7	0.0015	0.0024	0.0030	3.6	0.24	0.17	0.6
EPA REGION II	1,350	1,350	50.8250		0.24	0.034	30	0.0067	0.0108	0.0135	16.2	1.08	0.74	2.5
III	1,250	1,250	47.3798		0.23	0.031	28	0.0063	0.0100	0.0125	15.0	1.00	0.69	2.4
VI	1,000	1,000	37.478		0.18	0.025	22	0.0050	0.0080	0.0100	12.0	0.80	0.55	1.9
VII	110	110	3.6755		0.02	0.003	2	0.0005	0.0009	0.0011	1.3	0.09	0.06	0.2
VIII	930	930	34.4655		0.16	0.024	20	0.0047	0.0075	0.0093	11.1	0.75	0.51	1.8
IX	650	650	24.0310		0.12	0.016	14	0.0033	0.0052	0.0065	7.8	0.52	0.36	1.2
X	300	300	11.6749		0.05	0.008	7	0.0015	0.0024	0.0030	3.6	0.24	0.17	0.6
NATIONAL TOTAL	5,590	5,590	209.4297		1.00	0.141	123	0.0280	0.0448	0.0559	67.0	4.48	3.08	10.6

(A) MOST OF THESE SLUDGES ARE DISPOSED PERMANENTLY IN LARGE LAGOONS OR TAILINGS PONDS.

*MULTIPLY BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 8-b
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY COPPER ELECTROLYTIC REFINERS TOTAL SLUDGE - 1977 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENT (B)									
					Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	1,180	1,180	44.2	ALL TO	0.21	0.030	26	0.0059	0.0094	0.0118	14.2	0.94	0.65	2.2
NEW YORK	170	170	6.6	LAGOONS	0.03	0.004	4	0.0008	0.0014	0.0017	2.0	0.14	0.09	0.3
MARYLAND	1,250	1,250	47.4	(A)	0.23	0.031	28	0.0063	0.0100	0.0125	15.0	1.00	0.69	2.4
TEXAS	2,000	2,000	74.9		0.36	0.050	44	0.0100	0.0160	0.0200	24.0	1.60	1.10	3.8
MISSOURI	110	110	3.7		0.02	0.003	2	0.0005	0.0009	0.0011	1.3	0.09	0.06	0.2
MONTANA	460	460	17.1		0.08	0.012	10	0.0023	0.0037	0.0046	5.5	0.37	0.25	0.9
UTAH	470	470	17.2		0.08	0.012	10	0.0024	0.0038	0.0047	5.6	0.38	0.26	0.9
ARIZONA	715	715	26.4		0.13	0.018	15	0.0036	0.0057	0.0072	8.6	0.57	0.40	1.3
WASHINGTON	300	300	11.7		0.05	0.008	7	0.0015	0.0024	0.0030	3.6	0.24	0.17	0.6
EPA REGION II	1,350	1,350	50.8		0.24	0.034	30	0.0067	0.0108	0.0135	16.2	1.08	0.74	2.5
III	1,250	1,250	47.4		0.23	0.031	28	0.0063	0.0100	0.0125	15.0	1.00	0.69	2.4
VI	2,000	2,000	74.9		0.36	0.050	44	0.0100	0.0160	0.0200	24.0	1.60	1.10	3.8
VII	110	110	3.7		0.02	0.003	2	0.0005	0.0009	0.0011	1.3	0.09	0.06	0.2
VIII	930	930	34.3		0.16	0.024	20	0.0047	0.0075	0.0093	11.1	0.75	0.51	1.8
IX	715	715	26.4		0.13	0.018	15	0.0036	0.0057	0.0072	8.6	0.57	0.40	1.3
X	300	300	11.7		0.05	0.008	7	0.0015	0.0024	0.0030	3.6	0.24	0.17	0.6
NATIONAL TOTAL	6,655	6,655	249.2		1.19	0.168	146	0.0333	0.0533	0.0666	79.8	5.33	3.67	12.6

(A) MOST OF THESE SLUDGES ARE DISPOSED PERMANENTLY IN LARGE LAGOONS OR TAILINGS PONDS.

*MULTIPLY BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 8-c
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY COPPER ELECTROLYTIC REFINERS TOTAL SLUDGE - 1983 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENT (B)									
					Cd	Cr	Cu	Hg	Mn	Ni	Pb	Sb	Se	Zn
NEW JERSEY	1,215	1,215	45.6	ALL TO	0.22	0.031	27	0.0061	0.0097	0.0122	14.6	0.97	0.67	2.3
NEW YORK	175	175	6.8	LAGOONS	0.03	0.004	4	0.0008	0.0014	0.0018	2.1	0.14	0.09	0.3
MARYLAND	1,288	1,288	48.8	(A)	0.24	0.032	29	0.0065	0.0103	0.0129	15.5	1.03	0.71	2.5
TEXAS	2,060	2,060	77.2	↓	0.37	0.052	45	0.0103	0.0165	0.0206	24.7	1.65	1.13	3.9
MISSOURI	113	113	3.8		0.02	0.003	2	0.0005	0.0009	0.0011	1.3	0.09	0.06	0.2
MONTANA	474	474	17.6		0.08	0.012	10	0.0024	0.0038	0.0047	5.7	0.38	0.26	0.9
UTAH	484	484	17.8		0.08	0.012	10	0.0025	0.0039	0.0048	5.8	0.39	0.27	0.9
ARIZONA	819	819	30.3		0.15	0.020	18	0.0042	0.0066	0.0082	9.8	0.66	0.45	1.5
WASHINGTON	309	309	12.0		0.05	0.008	7	0.0015	0.0025	0.0031	3.7	0.25	0.18	0.6
EPA REGION II	1,390	1,390	52.3		0.25	0.035	31	0.0069	0.0111	0.0140	16.7	1.11	0.76	2.6
III	1,288	1,288	48.8		0.24	0.032	29	0.0065	0.0103	0.0129	15.5	1.03	0.71	2.5
VI	2,060	2,060	77.2		0.37	0.052	45	0.0103	0.0165	0.0206	24.7	1.65	1.13	3.9
VII	113	113	3.8		0.02	0.003	2	0.0005	0.0009	0.0011	1.3	0.09	0.06	0.2
VIII	958	958	35.4		0.16	0.024	20	0.0049	0.0077	0.0095	11.5	0.77	0.53	0.8
IX	819	819	30.3		0.15	0.020	18	0.0042	0.0066	0.0082	9.8	0.66	0.45	0.5
X	309	309	12.0		0.05	0.008	7	0.0015	0.0025	0.0031	3.7	0.25	0.18	0.6
NATIONAL TOTAL	6,937	6,937	259.8		1.24	0.174	152	0.0348	0.0556	0.0694	83.2	5.56	3.82	13.1

(A) MOST OF THESE SLUDGES ARE DISPOSED PERMANENTLY IN LARGE LAGOONS OR TAILINGS PONDS.

*MULTIPLY BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Dusts from smelting and fire refining are generated at a much lower annual rate than sludges. In contrast to sludges, dusts are eventually recycled, albeit after storage on the ground for periods of months. There are no dusts generated in electrolytic refining.

Arizona generates the largest amount of land disposed and hazardous wastes from smelting and fire refining by virtue of the fact that it has the most (7) smelters and fire refiners. On the other hand New Jersey and Maryland generates the largest quantity of electrolytic sludges because of the large production capacities of these states for electrolytic copper.

Projections of waste quantities for 1977 were based on additional electrolytic plant capacity now under construction in Arizona (27 MT/yr) and Texas (380 MT/yr) as given in the January 1975 Engineering and Mining Journal. By 1983 it appears that plant effluents from primary copper smelters and refiners will be limed, thereby increasing the quantities of sludge per metric ton of product by an estimated 26% for smelters and 3% for electrolytic refiners. Thus, sludge generation factors were increased from 19.7 kg and 2.4 kg/MT, respectively. An additional planned electrolytic refinery capacity of 36 MT/yr in Arizona and a 91 MT/yr smelter capacity increase in New Mexico was also considered in 1983 projections.

1.3 TREATMENT AND DISPOSAL TECHNOLOGY

This section contains descriptions of methods presently used for treatment and disposal of waste residuals on land and technology which is considered for adequate health and environmental protection.

1.3.1 Current Waste Treatment And Disposal Practices

Smelting and Fire Refining. Currently reverbatory slags are disposed in open dumps. This is adequate disposal since leaching is not evident.

Acid plant sludges are stored on land before recycle. Overflow from acid plant blowdown and miscellaneous slurries from scrubbers, cooling of anodes, washing etc. are settled in lagoons or tailings ponds. Disposal of sludges and slurries as currently done is inadequate due to the danger of toxic metal leaching.

Electrolytic Refining. Slurries from electrolytic copper refining are currently settled in unlined lagoons or tailings ponds and may be dredged and disposed of on land. These practices are inadequate due to the danger of toxic metal leaching.

The following paragraphs describe treatment and disposal technology levels for those wastes which have been ascertained as potentially hazardous.

1.3.2 Present Treatment and Disposal Technology (Level I)

Smelting and Fire Refining. Acid plant blowdown results in sludge which is dredged from settling lagoons and stored on land before eventual recycle to the reverbatory furnace. Overflow from acid plant settling lagoons is sent to mill tailings ponds. Miscellaneous slurries from scrubbers, contact cooling of anodes, washing, etc. are settled in lagoons or tailings ponds. They are either dredged, stored on land and eventually recycled or left permanently in tailings ponds. Dusts are stored on land for periods of months before recycling for reclamation of metallic values. The relative amounts of stored and disposed wastes vary from location to location and in response to market fluctuations.

Electrolytic Refining. At the present time miscellaneous slurries from electrolytic copper refining including spent acid, cooling water blowdown, scrubber water and washings are settled in an unlined lagoon or tailings pond. Settled sludges may either be dredged and disposed on land, or left permanently in tailings ponds.

1.3.3 Best Technology Currently Employed (Level II)

Smelting and Fire Refining. Lined lagoons are not used presently by the industry and thus Level II technology is unlined lagoons for acid plant blowdown and miscellaneous slurries. Land areas for storage or permanent disposal of dredged sludges (i.e. acid plant, miscellaneous) are not sealed. Therefore Level II technology is storage or permanent disposal on unsealed ground areas.

Electrolytic Refining. Lined lagoons are not used at present by the industry and thus Level II technology is unlined lagoons. Land areas for storage or permanent disposal of dredged sludges are not presently treated. Therefore Level II technology is storage or permanent disposal on untreated ground areas.

1.3.4 Technology To Provide Adequate Health And Environmental Protection (Level III)

Smelting and Fine Refining. Solubility tests as previously described indicate that hazardous constituents such as lead, chromium, antimony, cadmium, and copper are significantly leached from sludges which are stored in lagoons (i.e. acid plant sludges, scrubber sludges, slurries, etc) Lagoons should therefore be lined to prevent leaching and subsequent percolation through permeable soils to underlying water tables.

Instead of storing dredged sludge on land prior to recycling they should be stored in concrete lined pits to eliminate leaching. Alternatively, immediate recycle of sludge without storage may be feasible. Immediate recycle of dusts without ground storage precludes possible leaching and would be Level III technology.

Electrolytic Refining. Significant leaching of hazardous constituents (e.g. lead, copper, chromium) from electrolytic refining lagoon sediments is possible. The use of lined lagoons for permanent storage of such sediments should be practiced in permeable soils with underlying aquifers.

Tables 9 and 10 summarizes features of levels I, II, and III treatment and disposal technology for smelting and refining including assessment of the hazardous nature of residuals and adequacy of treatment and disposal.

1.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section.

1.4.1 Cost of Present Treatment And Disposal Technology (Level I)

The costs associated with the usual treatment and disposal of residuals from typical smelting and refining plants are presented below.

Smelting and Fire Refining. The plant has an annual capacity of 100,000 MT operating 350 days. Production of blister copper and fire refined copper results in a number of different potentially hazardous waste streams, namely wet sludges and slurries, and dry dusts.

Slurry from lime treatment of SO_2 gases from the acid plant amounts to 2,270 m^3/day and is processed through a thickener. The overflow from the thickener, 2200 m^3/day, enters a lagoon (Lagoon A). The daily inflow contains 0.77 MT of solids which form a sludge with an estimated density of 1200 kg/m^3 at the bottom of the lagoon. This results in a daily accumulation of 0.65 m^3 of sludge or 227.5 m^3/year. The lagoon size provides for 4 days of residency.

Miscellaneous slurries arising from sources such as wet scrubbers and anode cooling are deposited into a separate lagoon, designated as Lagoon B. One large lagoon could be used if desired. Its estimated daily inflow is 13,715 m^3 containing 4.8 MT of suspended solids. The sludge formed is assumed to have the same density as that in Lagoon A resulting in the formation of 4 m^3 of sludge/day and 1,400 m^3/yr. The lagoon size is based on 4 days of inflow.

Table 9

Treatment and Disposal Technology Levels

Smelting and Refining Category PRIMARY COPPER (Smelters and Fire Refiners) SIC 3331

MISC. DUSTS (Reverbatory furnace, roaster, converter)

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Fine, flour-like; oxides, sulfates, silicates, trace metals (As, Cd, Cu, Zn, Pb, Se)	Same as Level I	Same as Level I
Amount of Waste (kg/MT Product)	Dust - 17 kg	Same as Level I	Same as Level I
Factors Affecting Hazardousness	Contain heavy metals which may leach	No hazard since immediate recycle	No hazard since immediate recycle
Treatment/ Disposal Technology	Stored on land	Immediate recycle	Immediate recycle
Estimate of # + % of Plants Using Technology	(100%	Dust - Not known Recycle (at least one plant)	Same as Level II

Table 9-(cont'd.)

MISC. DUSTS - continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Dust - not adequate if heavy metals are leached	Adequate	Adequate
Problems and Comments	Leaching	-	-
Non-Land Environmental Impact	Possible contamination of ground and surface water	None	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Ground and surface water monitoring
Energy Requirements	None	None	None

Table 10

Treatment and Disposal Technology Levels

Smelting and Refining Category PRIMARY COPPER SIC 3331

Acid Plant sludges, Miscellaneous slurries (Smelters, Fire Refiners, Electrolytic Refiners)

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Acid Plant-Calcium sulfate and calcium sulfite with small amounts of heavy metals Other-Oxides, sulfates, silicates with heavy metals	Same as Level I	Same as Level I
Amount of Waste (kg/MT Product)	Acid plant sludge - 207 kg Misc. sludge* (smelter and fire ref) 17 kg Combined slurries** (Elec. ref.) 2.4 kg	Same as Level I	Same as Level I
Factors Affecting Hazardousness	Contain heavy metals which may leach (As, Pb, Cu, Hg, Sb, Zn, Cd, Cr, others)	Same as Level I	Same as Level I
Treatment/ Disposal Technology	Unlined lagoons; land storage or open dumping of dredged sludge (none is prevalent)	Unlined lagoons; land storage or open dumping of dredged sludge	Lined Lagoons; immediate recycle of sludge; storage in concrete pits prior to recycle or disposal in sealed soil areas
Estimate of # + % of Plants Using Technology	42 (100%)	42 (100%)	0 (0%)

* - Scrubbers, contact cooling, washing.

** - Spent acid, cooling water, scrubbers, washing

Table 10 - (cont'd.)

Acid Plant sludges, etc. (continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if heavy metals are leached	Inadequate if heavy metals are leached	Adequate
Problems and Comments	Particulates are fine sized. Solubility tests indicated significant heavy metal leaching potential (Cd, Cu, Pb, Zn)	Same as I	-
Non-Land Environmental Impact	Runoff could affect surface water quality; leaching could affect ground and surface water quality	Same as Level I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Periodic check to assure immediate recycle - non-land storage
Energy Requirements	Negligible	Negligible	Negligible

Characteristics of both lagoons necessary to provide retention for adequate settling are given below.

	Lagoon A	Lagoon B
Volume	8,800 m^3	55,000 m^3
Bottom width	34 m	91 m
Top width	46 m	103 m
Bottom length	67 m	182 m
Top length	79 m	194 m
Total depth	3 m	3 m
Depth of excavation	1.2 m	.65 m
Circumference	262 m	607 m
Dike Volume	3,117 m^3	10,987 m^2
Dike Surface	3,555 m^2	8,987 m^2
Total Width	59 m	119 m
Total length	93 m	210 m
Required area	.55 ha	2.5 ha

Two test holes are drilled for Lagoon A, four for Lagoon B. Both are located on rural land. The overflow from both lagoons go to a tailings pond which is part of the mine-mill-smelter complex. No significant portion of the tailings pond cost is attributable to smelter waste disposal because of insignificant volumes of smelter water wastes compared to mill tailing waste water (i.e. water from ore beneficiation).

Both lagoons are dredged twice/year requiring about 65 hours of dragline time. The dredged material is piled next to the lagoons, allowed to dry and then recycled. No provisions exist to prevent run-off or leaching of the dredged material for Level I.

Large amounts of dust are collected from each of the primary process operations. About 150 MT are stored on land near the smelter at any given time for subsequent recycling. No measures to prevent leaching are taken under Level I.

Cost of Level I treatment and disposal technology are given in Table 11.

Electrolytic Refining. The plant capacity is 160,000 MT operating 350 days/yr. The only land disposed waste is a combined dilute slurry composed of effluents from operations such as contact cooling of copper cast anodes, spent anode washing and plant washdown.

The daily flow is 6,800 m^3 which contains 210 kg of suspended solids. This flow enters a lagoon designed for 4 days settling time. Characteristics of the lagoon are:

TABLE 11

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY COPPER-SMELTING AND FIRE REFINING

CAPITAL COST

	Sludge
Lagoon A	
Site Preparation	
Survey	$ 345
Test Drilling	490
Sample Testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	4,145
Compacting	5,765
Fine Grading	1,600
Soil Poisoning	325
Transverse Drain Fields	545
Land	965
Lagoon B	
Site Preparation	
Survey	1,565
Test Drilling	980
Sample Testing	500
Report Preparation	1,500
Construction	
Excavation & Forming	14,615
Compacting	20,325
Fine Grading	4,045
Soil Poisoning	755
Transverse Drain Fields	3,285
Land	4,375
Equipment	
Dragline (5%)	3,500
Total	$71,075

TABLE 11 (cont.)

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY
PRIMARY COPPER-SMELTING AND FIRE REFINING

	ANNUAL COST
Land	$ 535
Amortization	
Construction	7,220
Equipment	555
Operating Personnel	790
Repair and Maintenance	
Construction	1,865
Equipment	175
Energy	
Fuel	75
Electricity	10
Taxes	135
Insurance	710
Total	$12,070

Volume	28,000 m^3
Bottom width	64 m
Top width	76 m
Bottom length	128 m
Top length	140 m
Total depth	3 m
Depth of excavation	.85 m
Circumference	443 m
Dike volume	6,948 m^3
Dike surface	6,335 m^2
Total width	90 m
Total length	155 m
Required area	1.4 ha

The lagoon is located on rural land.

The density of the sludge formed in the lagoon is 1340 kg/m^3. Only 55 m^3 of sludge is produced annually obviating dredging.

Costs of Level I treatment and disposal technology are given in Table 12.

1.4.2 Cost of Best Technology Currently Employed (Level II)

Smelting and Fire Refining. Collected dusts are recycled immediately without ground storage. This does not affect Level I costs. The technology and costs for sludge disposal are the same as Level I.

Electrolytic Refining. The technology and costs are the same as Level I.

1.4.3 Cost of Technology To Provide Adequate Health and Environmental Protection (Level III)

Smelting and Fire Refining. The lagoons are lined and the lagoon sludges are pumped to concrete pits for temporary storage prior to being recycled. The pits are sized to hold sludge formed over a 6 month period. Pumping the sludge eliminates the need for the dragline used in Level I for dredging.

The pits for Lagoon A and Lagoon B sludges are 5 x 5 x 8 m and 3 x 10 x 24 m, respectively. The same slurry pump is used to dredge the sludge from both lagoons. The slurry pump is required to operate 115 hrs/yr and 150 labor hours are allocated to this task. Incremental costs of Level III technology are listed in Table 13.

TABLE 12

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY
PRIMARY COPPER - ELECTROLYTIC REFINING

CAPITAL COST

	Sludge
Lagoon	
Site Preparation	
Survey	$ 875
Test Drilling	980
Sample Testing	500
Report Preparation	1,500
Construction	
Excavation & Forming	9,240
Compacting	12,855
Fine Grading	2,850
Soil Poisoning	550
Transverse Drain Fields	1,655
Land	2,450
Total	$33,455

ANNUAL COST

Land	$ 245
Amortization	
Construction	3,600
Equipment	-
Operating Personnel	-
Repair and Maintenance	
Construction	815
Energy	
Fuel	-
Electricity	30
Taxes	60
Insurance	335
Total	$ 5,085

TABLE 13

COST OF LEVEL III TREATMENT AND DISPOSAL TECHNOLOGY
PRIMARY COPPER - SMELTING AND FIRE REFINING

CAPITAL COST

	Sludge
Construction	
Lagoon A liner	$18,230
Lagoon B liner	88,565
Concrete Pit (Lagoon A)	8,380
Concrete Pit (Lagoon B)	24,535
Equipment	
Slurry Pump	13,730
Pipe, rigid (installed)	4,425
Pipe, flexible	880
(Dragline)	(3,500)
Total	$155,145

ANNUAL COST

Land	-
Amortization	
Construction	$ 16,195
Equipment*	2,470
Operating Personnel*	630
Repair and maintenance	
Construction	4,190
Equipment	775
Energy	
Fuel	-
Electricity*	(70)
Taxes	-
Insurance	1,550
Total	$25,740

* Costs shown are net costs, i.e. cost of new equipment less dragline associated costs which are no longer incurred.

Electrolytic Refining

Level III consists of lining the lagoon. The incremental costs are:

Capital cost

Lagoon liner	$ 48,880
Total	48,880

Annual cost

Construction Amortization	$ 5,675
Construction repair and maintenance	1,465
Insurance	490
Total	$ 7,630

Costs of Levels I, II and III treatment and disposal technology are summarized in Table 14 for scrubber sludge and dust from smelting and fire refining and in Table 15 for electrolytic plant sludges. Costs are given for individual waste streams (i.e. sludges, dusts) and for total waste streams in dollars per metric ton of dry and wet waste and copper product. The costs of treating each waste stream are also expressed as percentages of metal selling price.

Total industry costs for Levels I, II and III treatment and disposal technology have been estimated in 1973 dollars based on 1973 production and are presented in Table 14 for smelters and fire refiners and Table 15 for electrolytic refiners. The annualized cost for the primary copper smelting and fire refining industry is estimated as $190,000 for Levels I and II treatment and disposal technology and $6,130,000 for Level III technology (i.e. technology adequate for environmental protection). Levels I and II technology represents approximately 0.009% of 1973 national sales. The cost of Level III technology represents 0.28% of national sales of primary copper.

The annualized cost for the primary copper electrolytic refining industry is estimated as $50,000 for Levels I and II treatment and disposal technology. This represents approximately 0.002% of 1973 national sales. The annualized cost for Level III treatment and disposal technology is estimated as $120,000 or approximately 0.005% of national sales.

TABLE 14 COST SUMMARY FOR TREATMENT AND DISPOSAL TECHNOLOGIES: PRIMARY COPPER-SMELTING AND FIRE REFINING

Annual Production: Model Plant-100,000 MT Industry-1,621,131 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.02
Dust	0.0015

Cum. Unit Waste Disposal Costs:

Waste (Type)	Level I			Level II			Level III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$36.39	$14.56	$0.71	$36.39	$14.56	$0.71	$115.83	$46.33	$2.26
Annual Cost	6.18	2.47	0.12	6.18	2.47	0.12	19.36	7.74	3.78
Dust - Recycled	-		-	-		-	-		-
Total Capital Cost	-		$0.71	-		$0.71	-		$2.26
Total Annual Cost	-		0.12	-		0.12	-		3.78

Cum. Total Industry Waste Disposal Costs ($ Million):

Waste (Type)	Level I		Level II		Level III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$1.15	$0.19	$1.15	$0.19	$3.66	$6.13
Total	$1.15	$0.19	$1.15	$0.19	$3.66	$6.13

1973 Metal Price: $1,309/MT

Treatment Cost As Percentage of Metal Selling Price

Waste (Type)	Level I		Level II		Level III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.05%	0.01%	0.05%	0.01%	0.17%	0.29%
Total	0.05%	0.01%	0.05%	0.01%	0.17%	0.29%

TABLE 15 COST SUMMARY FOR TREATMENT AND DISPOSAL TECHNOLOGIES: PRIMARY COPPER-ELECTROLYTIC REFINING

Annual Production: Model Plant-160,000 MT Industry-1,542,368

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.0005

Cum. Unit Waste Disposal Costs:

	Level					
Waste (Type)	I		II		III	
	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.
Sludge						
Capital Cost	$453.93	$0.21	$453.93	$0.21	$1117.16	$0.51
Annual Cost	68.97	0.03	68.97	0.03	172.52	0.08
Total Capital Cost	-	$0.21	-	$0.21	-	$0.51
Total Annual Cost	-	0.03	-	$0.03	-	$0.08

Cum. Industry Waste Disposal Costs ($ Million):

	Level					
	I		II		III	
Waste (Type)	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$0.32	$0.15	$0.32	$0.05	$0.79	$0.12
Total	$0.32	$0.15	$0.32	$0.05	$0.79	$0.12

1973 Metal Price: $1,309/MT

Percent Treatment Cost/Price of Metric Ton of Production

	Level					
	I		II		III	
Waste (Type)	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.02%	0.002%	0.02%	0.002%	0.04%	0.006%
Total	0.02%	0.002%	0.02%	0.002%	0.04%	0.006%

2.0 PRIMARY SMELTING AND REFINING OF LEAD

2.1 INDUSTRY CHARACTERIZATION

There are only 7 primary lead smelting and refining plants in the United States owned by four different corporations. These are large operations however ranging in size from 56,000 MT to 209,000 MT per year. Two plants conduct smelting operations only; one plant conducts refining operations only and the remaining four plants smelt and refine lead ore concentrates. In 1973 total U.S. production of refined lead was 687,739 short tons and average selling price was $0.1629/lb (Ref. 2). The total U.S. sales are estimated as $224,065,000.

Features of individual lead smelters and refiners including capacities, presence of acid and slag fuming plants and gas cleaning methods are summarized in Table 16. All of these features have an influence on the types and quantities of residuals disposed or stored on land. Table 17 gives state by state, EPA region and national primary lead production capacities.

The ore from which primary lead is produced contains both lead and zinc. The lead concentrate is usually roasted in traveling-grate sintering machines, thereby removing sulfur and forming lead oxide. The lead oxide, sinter, coke, and flux (usually limestone) are fed to the blast furnace, in which oxide is reduced to metallic lead. The lead is further refined in reverbatory furnaces. By product recovery from lead refining residues include copper, zinc, antimony, arsenic, silver, gold, bismuth, cadmium, selenium, tellurium, nickel and cobalt. Process details are further described in the next section.

2.2 WASTE CHARACTERIZATION

This section contains descriptions of production technology at primary lead smelters and the resultant byproducts or wastes which are either recycled directly, shipped to other smelters for further metallic recovery or disposed of on land. Estimates are given for the quantities of wastes and potentially hazardous constituents thereof which are disposed of on land either in lagoons or open dumps.

2.2.1 Process Description

The hypothetical lead plant is assumed to have an annual capacity of 110,000 metric tons (MT) of 99.99 + % lead per year (121,000 short tons/year). Five of the six U.S. smelters have within ±15 percent of this capacity and the sixth has approximately double this capacity. Lead smelters are estimated to operate 350 days/year; the daily capacity is, therefore, 314 MT (346 short tons).

Table 16

CHARACTERISTICS OF U.S. LEAD SMELTERS

PLANT	YEAR FIRST OPERATED	ORIGIN OF CONCENTRATES	SLAG* FUMING PLANT	METHOD OF HANDLING SO_2 GAS FROM SINTER MACH.	REFINERY ADJACENT TO SMELTER	REFINED LEAD PRODUCTION 1000 UNITS/YEAR	
						MT	(Tons)
ST. JOE HERCULANEUM, MO.	1891	U.S.A. (S.E. MO. ORES)	NONE; ZINC DISCARDED IN SLAG	ACID PLANT	YES	209	(230)
AMAX BOSS, MO.	1968	U.S.A. (S.E. MO. ORES)	NONE; ZINC DISCARDED IN SLAG	ACID PLANT	YES	121	(133)
ASARCO GLOVER, MO.	1968	U.S.A.	NONE; ZINC DISCARDED IN SLAG	SPRAY CHAMBER, BAGHOUSE, STACK	YES	98	(108)
BUNKER HILL KELLOGG, IDA.	1917	U.S.A. CANADA, PERU, AUSTRALIA	YES	ACID PLANT	YES	121	(133)
ASARCO E. HELENA, MT	1888	U.S.A. CANADA, PERU, AUSTRALIA	YES	SPRAY CHAMBER, BAGHOUSE	NO; BULLION REFINED AT ASARCO REFINERY IN OMAHA	58	(64)
ASARCO EL PASO, TX	1887	U.S.A. CANADA, PERU, MEXICO	YES	SPRAY CHAMBER, BAGHOUSE, STACK	NO; BULLION REFINED AT ASARCO REFINERY IN OMAHA	56	(62)
ASARCO OMAHA, NE	1870	N.A.	N.A.	N.A.	N.A.	123	(136)

N.A. - NOT APPLICABLE

SOURCE:
DRAFT EFFLUENT
GUIDELINES, PRIMARY
LEAD SMELTING AND
REFINING, EPA, JUNE, 1974

*RECOVERY OF ZINC FROM LEAD SLAG

Table 17

GEOGRAPHICAL DISTRIBUTION OF PRIMARY LEAD SMELTING AND REFINING CAPACITY

STATE	NO. OF PLANTS	PLANT DISTRIBUTION BY CAPACITY (MT)		PLANT DISTRIBUTION BY PROCESS			CAPACITY PER PROCESS (MT)		
		50K-100K	100K-200K	S	SRP	RP	S	SRP	RP
IDAHO	1	0	1	0	1	0	0	113000	0
MISSOURI	3	1	2	0	3	0	0	372000	0
MONTANA	1	1	0	1	0	0	98000	0	0
NEBRASKA	1	0	1	0	0	1	0	0	163000
TEXAS	1	1	0	1	0	0	98000	0	0
EPA REGION									
I	0	0	0	0	0	0	0	0	0
II	0	0	0	0	0	0	0	0	0
III	0	0	0	0	0	0	0	0	0
IV	0	0	0	0	0	0	0	0	0
V	0	0	0	0	0	0	0	0	0
VI	1	1	0	1	0	0	98000	0	0
VII	4	1	3	0	3	1	0	372000	163000
VIII	1	1	0	1	0	0	98000	0	0
IX	0	0	0	0	0	0	0	0	0
X	1	0	1	0	1	0	0	113000	0
NATION	7	3	4	2	4	1	196000	485000	163000

S = SMELTING   SRP = SMELTING AND PYROREFINING   RP = PYROREFINING
SMELTING CAPACITY = LEAD BULLION/YEAR   REFINERY CAPACITY = REFINED LEAD/YEAR

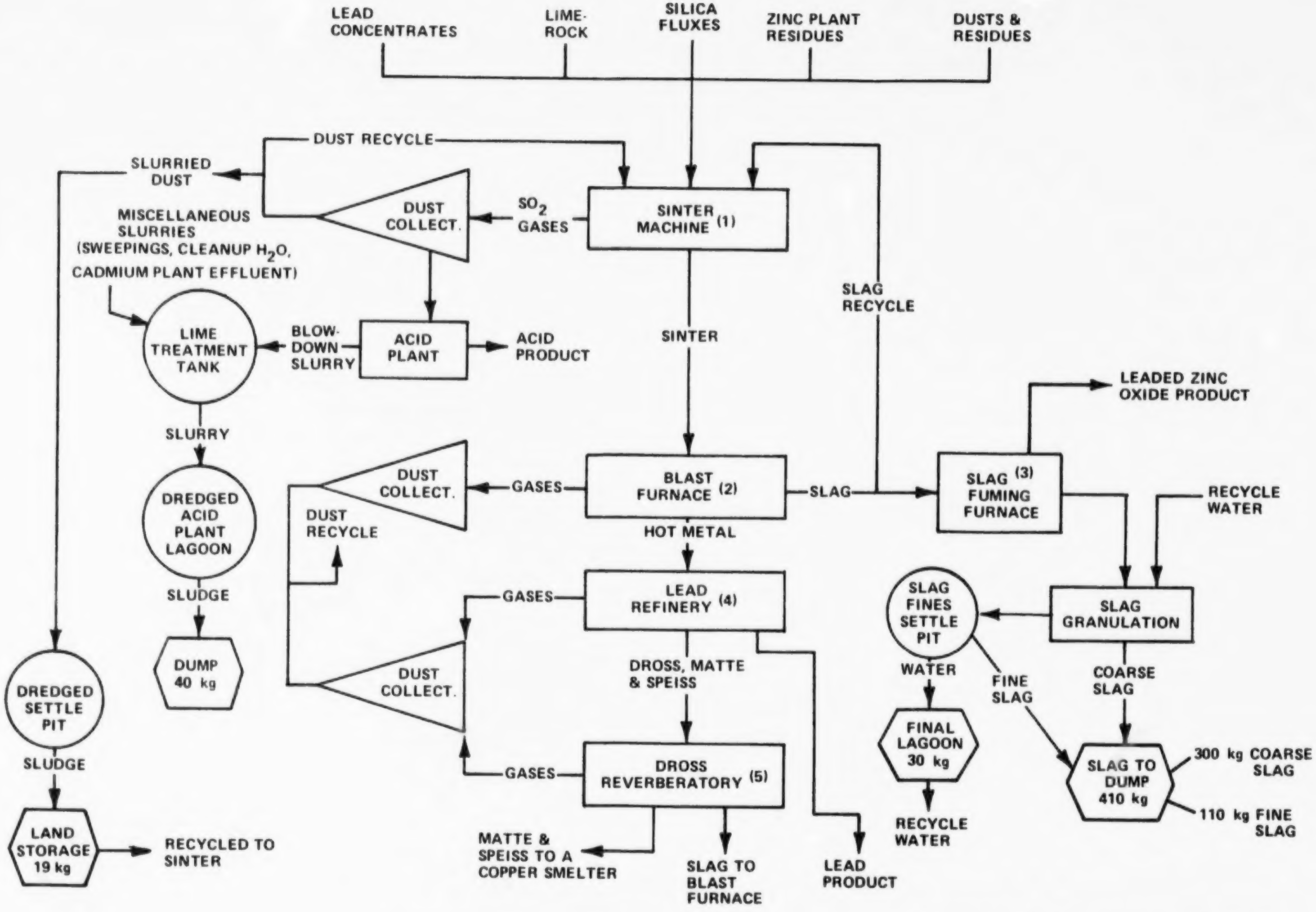
Input (feed) to the plant amounts to approximately 163,000 MT/year (180,000 short tons/year) of flotation concentrates from sulfide ores (and miscellaneous lead-bearing materials) averaging approximately 75 percent lead. In addition to concentrates, the feed may include lead-bearing dusts and fumes from gas cleaning operations, sinters, fines, slags, drosses, and residues generated in the lead smelter, and residues such as flue dusts shipped from copper and zinc smelters. This conglomeration of feed materials may contain such recoverable byproduct metals as zinc, silver, arsenic, antimony, bismuth, cadmium, copper, gold, platinum, palladium, nickel, colbalt, selenium, tellurium, and germanium.

A representative flow diagram is shown in Figure 3. This diagram shows the quantities of wastes or recyclable material either stored on land or permanently disposed in lagoons or open dumps. The principal unit smelting operations are (1) double-pass sintering to remove sulfur and simultaneously agglomerate fine-sized feed materials: (2) blast furnace smelting of sinter with coke and fluxes to make a lead bullion containing most of the byproduct metal values, and a slag containing most of the zinc and worthless gangue material; (3) slag fuming to recover byproduct zinc oxide, a lead fume for recycling, and a discardable waste slag; (4) drossing the molten impure bullion with air and sulfur to remove most of the copper; (5) reverberatory smelting of the copper dross to produce an enriched copper matte for shipment to a copper smelter, and a crude lead bullion for recycle to the drossing operation. Auxiliary smelter operations include (1) collecting dusts from the sintering, blast, and reverberatory furnace operations for reprocessing; (2) precipitating fumes from the same gas streams for diverse treatment to recovery their lead, zinc, cadmium, antimony, and arsenic contents; and (3) recovering sulfur dioxide from the cleaned sinter plant gases in a sulfuric acid plant.

Blast furnace slag is the primary solid waste produced at the smelter in large quantity. Smaller amounts of particulate sludges emanate from the acid plant and from other sources as will be detailed below.

After drossing (i.e. floating on surface of melt) to remove most of the copper, the crude lead bullion is refined as shown in Figure 3-a to produce marketable lead, silver, gold, bismuth, and reprocessible byproduct slags, fumes, solutions, and residues containing zinc, cadmium, antimony, arsenic, selenium, tellurium, copper, tin, zinc, nickel, and cobalt.

The main steps in the pyrometallurgical refining of lead bullion comprise (1) softening, using air to remove antimony, residual copper, arsenic, and tin; (2) desilverizing the softened lead bullion with metallic zinc; (3) dezincing and casting of pure lead. The oxidized dross, slag, and fume from the softening furnace are subsequently smelted


LEAD CONCENTRATES
LIME-ROCK
SILICA FLUXES
ZINC PLANT RESIDUES
DUSTS & RESIDUES
DUST RECYCLE
SLURRIED DUST
MISCELLANEOUS SLURRIES (SWEEPINGS, CLEANUP H_2O, CADMIUM PLANT EFFLUENT)
DUST COLLECT.
SO_2 GASES
SINTER MACHINE (1)
SLAG RECYCLE
LIME TREATMENT TANK
BLOW-DOWN SLURRY
ACID PLANT
ACID PRODUCT
SINTER
LEADED ZINC OXIDE PRODUCT
SLURRY
DREDGED ACID PLANT LAGOON
DUST COLLECT.
GASES
BLAST FURNACE (2)
SLAG
SLAG (3) FUMING FURNACE
RECYCLE WATER
DUST RECYCLE
HOT METAL
SLUDGE
GASES
LEAD REFINERY (4)
SLAG FINES SETTLE PIT
SLAG GRANULATION
DUMP 40 kg
DUST COLLECT.
DROSS, MATTE & SPEISS
WATER
FINE SLAG
COARSE SLAG
DREDGED SETTLE PIT
FINAL LAGOON 30 kg
SLUDGE
GASES
DROSS REVERBERATORY (5)
SLAG TO DUMP 410 kg
300 kg COARSE SLAG
110 kg FINE SLAG
LAND STORAGE 19 kg
RECYCLED TO SINTER
MATTE & SPEISS TO A COPPER SMELTER
SLAG TO BLAST FURNACE
LEAD PRODUCT
RECYCLE WATER

NUMERICAL VALUES ARE IN KILOGRAMS/METRIC TON LEAD PRODUCED

Figure 3 PRIMARY LEAD SMELTING AND REFINING

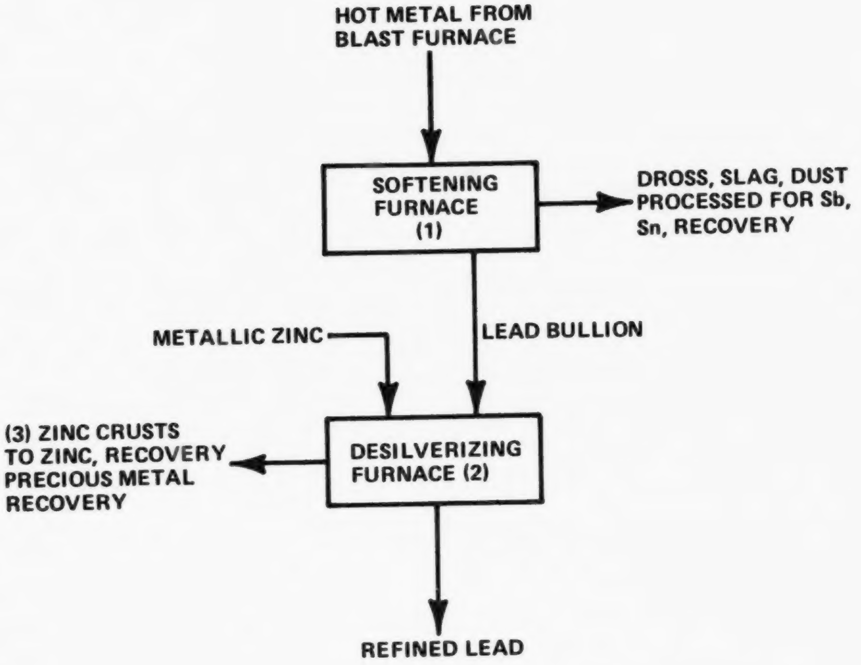

HOT METAL FROM BLAST FURNACE
SOFTENING FURNACE (1)
DROSS, SLAG, DUST PROCESSED FOR Sb, Sn, RECOVERY
LEAD BULLION
METALLIC ZINC
(3) ZINC CRUSTS TO ZINC, RECOVERY PRECIOUS METAL RECOVERY
DESILVERIZING FURNACE (2)
REFINED LEAD

Figure 3 PYROMETALLURGICAL LEAD REFINING

to produce marketable antimonial lead, antimony oxide, and tin metal or tin compounds and alloys. The silver-rich zinc crusts are retorted to remove and recover zinc for reuse and then cupelled to make a precious metal dore bullion that is subsequently parted chemically or electrolytically into marketable silver, gold, and platinum group metals.

Lead smelting and refining, although comprising numerous unit process steps to separate the many metal values in the smelter feed, is efficient. The practice of collecting, recycling, and reprocessing of dusts, fumes, slags, and residues, effectively reduces the loss of metal values. An estimated 95 percent of the principal metal values, including silver, are recovered either in marketable high-quality metals or in recoverable byproducts.

Gases from the first section of the sinter machine are high enough in SO_2 to permit sulfuric acid recovery by conventional technology. The amount of acid product is 86,000 MT (95,000 short tons) per year or 245 MT (270 short tons) per operating day.

2.2.2 Description of Waste Streams

a. Blast Furnace Slag

Molten siliceous slag from the blast furnace is 70% recycled to sinter, but 30% is passed thru a slag fuming furnace to remove zinc and some residual lead, then granulated by water quenching. The amount of granulated slag leaving the smelter and going to the main slag dump is 300 Kg/MT of lead product or 93 MT/day (103 short tons/day). Some smelters do not fume slag; untreated granulated slag then goes directly to the dump.

b. Slag Granulation Slurry

Slag granulating water amounts to approximately 370 cubic meter/day (97,000 gal/day). This water flows to two unlined pits where coarse solids settle out in the short residence time allowed. The pits, holding 150 m^3 (39,000 gal), are dredged periodically and the dredged fine slag is placed permanently on the main slag dump. This fine slag is considered to be closely similar in chemical analysis to the main granulated slag waste but is smaller in particle size. The amount dredged from the pits is 110 Kg/MT of lead product or 36 MT/day (40 short tons/day).

Overflow from the settling pit carrying suspended very fine slag particulates goes to a larger, longer residence time polishing lagoon from which water is recycled to slag granulation. The amount of these very fine settlings is estimated at 30 Kg/MT of lead product or 10 MT/day (11 short tons/day) and the size of the polishing lagoon is approximately 6000 cubic meter (1,600,000 gal).

This lagoon is dredged periodically and the dredgings are disposed of on the slag dump.

c. Slurries from Primary Gas-Cleaning Operations

Sinter gases laden with SO_2 are partially scrubbed of particulates before entering the acid plant. Gases from other primary operations such as sinter crushing and blast furnacing are similarly cleaned in electrostatic precipitators, wet scrubbers, and baghouses. The bulk of the dry dusts and wet slurries resulting from primary gas cleaning operations are recycled to sinter but a portion of the slurry containing perhaps 6.1 MT/day (6.7 short tons/day) of solids (in an unknown amount of water) is not immediately recycled. Gas cleaning practice is believed to vary widely from plant to plant.

Handling of the slurry is by settling in an unlined pit of capacity 260 cubic meters (68,000 gal). The pit is dredged periodically and the sludge is piled on the slag dump where it resides for several months before recycle to sinter. This sludge is produced at a rate of 19 Kg/MT of lead product. It is believed that this material may not be recycled at some plants, i.e., it is then definitely a solid waste disposed of on land.

d. Acid Plant Blowdown Slurry

Necessary acid plant blowdown and lime treatment of the resulting effluent produces approximately 230 cubic meter/day (60,000 gal/day) slurry effluent containing 6.3 MT/day (6.9 short tons/day) solids. This slurry goes to a lime treatment tank and then to the acid plant lagoon (unlined) of capacity approximately 6,800 cubic meters (240,000 cubic feet). Miscellaneous slurries (see item e. below) join the limed acid plant blowdown in this lagoon. The lagoon is dredged periodically and the sludge is piled on the ground to dry before eventual recycle to sinter. The amount of dry dredgings attributable to the acid plant blowdown is as estimated above, i.e., 20 Kg/MT of lead product or 6.3 MT/day (6.9 short tons/day). This material may not be recycled at some plants.

e. Miscellaneous Slurries

Miscellaneous dusts such as sweepings are slurried and combined with wastewaters such as cleanup water and cadmium plant effluent; this effluent is lime treated and settled in the above-mentioned acid plant lagoon. The quantity of miscellaneous slurries entering the lagoon is estimated to be less than 230 cubic meter/day (60,000 gal/day) and the contained solids are estimated to be less than 20 Kg/MT of lead product or 6.3 MT/day (6.9 short tons/day). The settled sludge is dredged and dried on the ground in mixture with the above-described acid plant sludge. Total dry sludge per day from acid plant blowdown and miscellaneous slurries is 40 Kg/MT lead product.

f. Dry Dusts From All Gas Cleaning Operations

A portion of the dusts collected from sintering, blast furnacing, and other operations in baghouses and other dry dust collectors is not slurried but is recycled in dry condition to the sinter machine. It is estimated that 19,000 MT/year (20,900 short tons/year) are handled in this manner from the typical plant. Recycle is immediate and no solid waste disposal results. Approximately 2,000 MT or 10% of dusts are land stored before recycle.

Chemical analyses of waste samples collected from various primary lead smelters are given in Appendix A. In solubility tests described in Appendix B it was found that dredged sludge originating from the sinter dust scrubbers and acid plant leached significant concentrations of cadmium, copper, lead and zinc. This may be expected from the high concentrations of heavy metals in these wastes and fine particulate sizes. Because of evidence indicating significant leaching of heavy metals from sinter plant, acid plant and miscellaneous sludges, these wastes are considered potentially hazardous.

The granulation of lead fuming slag comminutes the slag to sand and pebble size separates. Solubilization of copper, antimony and zinc in the order of 1 ppm was observed in leaching studies (see Appendix B). This degree of solubilization was not considered sufficient to identify the lead fuming slag as a potentially hazardous waste at this time.

2.2.3 Waste Quantities

Table 18 summarizes the generation factors for each of the land disposed or stored residuals in the lead smelting and refining category. This table also gives typical concentrations of potentially hazardous constituents. Table 19 gives the yearly quantity of residuals and potentially hazardous constituents for a typical plant producing 110,000 MT (121,000 short tons) of 99.99% price lead per year.

Using the waste generation factors and concentration factors contained in Table 18 and lead smelter and refining plant capacities, estimates have been made for the quantities of land disposed or stored waste and hazardous constituents thereof. These estimates are given for 1974, 1977, and 1983 on a state-by-state, regional and national level in Table 20.

The state in which the largest quantity of potentially hazardous waste (i.e. sludge) is generated is Missouri which is the largest producer of primary lead (3 of the 7 primary lead smelters are in Missouri). Idaho and Texas also generate significant quantities of potentially hazardous waste from lead smelting.

TABLE 18 WASTE RESIDUAL FACTORS, PRIMARY LEAD SMELTING AND REFINING

Type of Waste	Generation Factor Kg/MT	Concentration Factor (ppm)							
		Cd	Cr	Cu	Hg	Mn	Pb	Sb	Zn
Slag	410	166	166	1720	-	1090	31,100	59	72,300
Sludge	89	6900	27	5820	180	925	143,500	924	79,900

TABLE 19 YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL PLANT, PRIMARY LEAD SMELTING AND REFINING

Type of Waste	Total Waste Quantity (MT)	Quantity of Potentially Hazardous Constituents (MT)							
		Cd	Cr	Cu	Hg	Mn	Pb	Sb	Zn
Slag	45,100	7.4	7.4	78	-	49	1400	2.7	3260
Sludge	9,790	68	0.26	57	1.8	9.1	1400	9.1	780

*Quantities calculated from generation and concentration factors given in Table 18 based on annual production at 110,000 MT. Divide by 350 to obtain daily quantities, multiply by 1.1 to convert to short tons.

Table 20-a

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY LEAD SMELTING AND REFINING TOTAL SLUDGE, 1974, 1977, 1983 (METRIC TONS, DRY WEIGHTS*)

STATE	TOTAL DISPOSAL	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS							
					Cd	Cr	Cu	Hg	Mn	Pb	Sb	Zn
MISSOURI	13,100	4,500	1,010	LAGOON	3.6	0.1	27	0.7	0.7	630	0.4	350
		8,600 (A)	2,150	OPEN DUMP	120	0.1	46		1.0	1270	1.0	710
IDAHO	3,300	3,300 (B)	760	LAGOON	2.1	0.2	21	1.7	9.6	460	9.9	260
MONTANA	150	150 (B)	30	LAGOON	0.1	-0-	1.6	-0-	-0-	25	-0-	3.8
TEXAS	1,800	1,800 (B)	420	LAGOON	1.2	0.1	11	0.9	5.3	250	5.4	150
NEBRASKA	NOT KNOWN (C)	NOT KNOWN	NOT KNOWN	LAGOON	← NOT KNOWN →							
		D		OPEN DUMP								
EPA REGION												
VI	1,800	1,800	420		1.2	0.1	11	0.9	5.3	250	5.4	150
VII	13,100	13,100	3,160		124	0.2	73	0.7	1.7	1900	1.4	1060
VIII	150	150	30		0.1	-0-	1.6	-0-	-0-	25	-0-	3.8
X	3,300	3,300	760		2.1	0.2	21	1.7	9.6	460	9.9	260
NATIONAL TOTAL	18,400	18,400	4,400 (B)		127	0.5	107	3.3	17	2640	17	1470

NOTES:

(A) THIS RESIDUE IS A DRY DUST RATHER THAN A SLUDGE, AND IS STORED ON THE GROUND FOR RECYCLE.

(B) A HIGH PERCENTAGE OF ALL SLUDGES IS DREDGED FROM LAGOONS, DRIED AND STORED ON LAND, AND EVENTUALLY RECYCLED. A LOW PERCENTAGE IS PERMANENTLY LAND-DISPOSED IN OPEN DUMPS OR LARGE TAILINGS PONDS.

(C) PLANT IN NEBRASKA IS REFINERY WITHOUT SMELTER. THERE IS THEREFORE NO GYPSUM SLUDGE OR SLAG WASTE. THERE IS SMALL BUT UNKNOWN QUANTITY OF SLUDGE ASSOCIATED WITH WATER POLLUTION CONTROL. NO DATA AVAILABLE ON AMOUNT.

* INCLUDES GYPSUM SLUDGE FROM ACID PLANT AND SLURRIED DUST FROM SINTER.

MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORP.

Table 20-b
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY LEAD SMELTING AND REFINING TOTAL SLAG - 1974, 1977, 1983 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	DISPOSAL METHOD	CONSTITUENTS						
				Cd	Cr	Cu	Mn	Pb	Sb	Zn
MISSOURI	146,000	0	OPEN DUMP	83	30	330	150	6,800	15	17,800
IDAHO	153,000			1.5	23	230	2,200	3,800	5	6,400
MONTANA	111,000			1.1	17	170	1,600	2,800	3.7	4,700
TEXAS	107,000			1.0	16	160	1,600	2,700	3.5	4,500
NEBRASKA	7,100			-0-	1.1	11	100	180	0.2	300
EPA REGION										
VI	107,000			1.0	16	160	1,600	2,700	3.5	4,500
VII	153,000			83	31	340	250	7,000	19	22,300
VIII	111,000			1.1	17	170	1,600	2,800	3.7	4,700
X	153,000			1.5	23	230	2,200	3,800	5	6,400
NATIONAL TOTAL	524,000 **			87	87	900	5,700	16,300	31	37,900

*THIS WASTE BELIEVED NONHAZARDOUS BASED ON CALSPAN SOLUBILITY TESTS DESCRIBED IN APPENDIX B.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

** APPROXIMATELY ONE-THIRD OF TOTAL SLAG IS A FINE-PARTICLE FRACTION SETTLED FROM SLAG GRANULATING WATER, DREDGED, AND ULTIMATELY PLACED ON OPEN DUMPS.

SOURCE: CALSPAN CORPORATION

There is no documented, planned plant expansion in the industry or anticipated changes in production or pollution control technology which would affect the quantities of land disposed wastes. Therefore, estimates for 1977 and 1983 are the same as those for 1974.

2.3 TREATMENT AND DISPOSAL TECHNOLOGY

2.3.1 Current Waste Treatment and Disposal Practices

Currently, discard slag is open dumped. Acid plant and sinter plant sludges are settled in unlined lagoons, dredged and either stored or disposed of permanently in unsealed ground areas. As discussed previously, slag is not considered hazardous at this time. Sludge is susceptible to heavy metal leaching and levels of technology for its treatment and disposition are discussed in the following sections.

2.3.2 Present Treatment and Disposal Technology (Level I)

At the present time lagoons are unlined. Dredged acid plant and sinter plant sludge are either permanently disposed of on unsealed ground areas or stored on unsealed ground areas. Potentially hazardous metals including lead, cadmium, copper, and zinc may be leached from sludges through porous soils to underlying aquifers. Present practices need augmentation.

2.3.3 Best Technology Currently Employed (Level II)

At one or more primary lead smelters lime sludge from the acid plant and scrubber water sludge from the sinter plant are immediately recycled thereby precluding land storage or disposal. This practice is considered Level II technology.

2.3.4 Technology To Provide Adequate Health and Environmental Protection (Level III)

The practice of immediate recycle of acid plant and sinter scrubwater sludges would qualify as Level III technology. In the event that sludges cannot be immediately recycled, concrete pits and lined lagoons will be required as Level III technology to prevent leaching of metals including cadmium, copper, lead and zinc through permeable soils into aquifers. Sludges originating from the sinter plant or the acid plant which are permanently disposed of on land should be chemically fixed (i.e. solidified and immobilized) before ground disposal as Level III technology to prevent leaching through permeable soils to aquifers.

Table 21 summarizes features of Levels I, II, and III treatment disposal technology including assessment of the hazardous nature of residuals and the methods and adequacy of treatment and disposal methods for sludges.

2.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. The hypothetical plant is assumed to have an annual capacity of 110,000 MT of lead product per year and operate 350 days per year.

2.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Slag granulation water, slurries from primary gas cleaning, acid blowdown and miscellaneous activities are directed into a lagoon. Daily inflow is 1,480 m^3 containing 23.8 MT of solids. The postulated lagoon size is 6,000 m^3. Characteristics of the lagoon to accommodate the daily volume of wastewater and approximately 200 days of sludge accumulation are:

Bottom width	36 m
Top width	44 m
Bottom length	72 m
Top length	80 m
Total depth	2 m
Depth of excavation	.7 m
Circumference	259 m
Dike volume	1,888 m^3
Dike surface	2,671 m^2
Total width	55 m
Total length	91 m
Required area	.5 ha

Two test holes and 4 analyses of soil samples to ascertain lagoon structural stability are included in site preparation. The lagoon is situated on rural land.

The daily inflow of 23.8 MT of solids are estimated to form 14.3 m^3 of sludge in the lagoon. The lagoon is dredged when one-half filled with sludge. This occurs every 209 days or 1.7 times/operating year. The sludge, amounting to 5,100 m^3/yr, is allowed to dry next to the lagoon and hauled to an on-site slag dump. Dredging, loading and hauling require about 190 hrs/yr of dragline and 190 hrs/yr of front loader and truck time.

Table 21

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary Lead SIC 3332

Sludges (sinter plant and acid plant)

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Sinter plant - colloidal and silt size aggregates-calcium silicates and oxides predominant acid plant - colloidal and silt size aggregates - predominantly calcium sulfates and sulfites	Same as I	Same as I
Amount of Waste (kg/MT Product)	Sinter plant sludge - 19 kg Acid plant sludge - 40 kg	None if recycled	None if recycled 59 kg, if not
Factors Affecting Hazardousness	Contains heavy metals including As, Pb, Cu, Zn, Cr, Sb which may leach	No hazard, if recycled	No hazard if recycled. Same as I, if lagoons or sealed ground leaks
Treatment/ Disposal Technology	Unlined settling pits and lagoons with land storage or disposal of sludge	Immediate recycle to sinter	Immediate recycle to sinter or concrete settling pits, lined lagoons and chemical fixation of land disposed sludge
Estimate of # + % of Plants Using Technology	Predominant 4-6 plants (60-85%)	At least one (15%)	At least one (15%) has immediate recycle. Concrete pits, lined lagoons and sludge fixation not presently done.

Table 21-(cont'd.)

SLUDGES (continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if leaching of heavy metals	Adequate	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	Possible contamination of ground and surface water if run off or leaching of heavy metals	May interfere with air pollution controls on sinter by increasing emissions	May interfere with air pollution control on sinter if recycled by increasing emissions
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None required	Ground water monitoring wells
Energy Requirements	Negligible	Increased energy for sinter recycle (fuel use for agglomeration)	Same as II

The slurry from gas cleaning operations is passed through an unlined pit with a capacity of 260 m^3 (2 x 5.2 x 15.6). The slurry contains about 6.1 MT/day of solids, 4.1 MT (67%) of which are estimated to settle in the pit. (The solids form a sludge with an estimated density of 1650 Kg/m^3, and about 3 m^3 of sludge accumulate daily).

Dredging, loading and hauling of sludge from the slag granulation pits is estimated to require an average of 2 hr/day of backhoe and frontloader and 1 hr/day of truck time. Comparable requirements for the gas cleaning slurry pit are 60 hr/yr of backhoe and front loader time and 30 hr/yr of truck time. The costs of Level I sludge disposal are given in Table 22.

2.4.2 Cost of Best Technology Currently Employed (Level II)

The only change from Level I is the immediate recycling of wastes dredged from the gas cleaning slurry pit. Costs remain as in Level I.

2.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection (Level III)

The lagoon is lined; dredgings from the lagoon are chemically fixed prior to being hauled to the slag dump.

Additional costs are as follows:

Capital Costs	
Lagoon liner	$ 17,575
Total	$ 17,575
Annual Costs	
Construction Amortization	$ 2,040
Construction Maintenance and Repair	525
Chem-Fix of sludge	67,320
Insurance	175
Total	$ 70,060

TABLE 22 COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY LEAD PLANT

CAPITAL COST

	Sludge
Lagoon	
Site Preparation	
Survey	$ 315
Test Drilling	490
Sample Testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	2,510
Compacting	3,495
Fine Grading	1,200
Soil Poisoning	320
Transverse Drain Fields	505
Land	875
Sump Construction	
Slag Granulation Pits (2)	400
Primary Gas Cleaning	345
Equipment	
Dragline (10%)	7,000
Dump Truck (70%)	7,875
Front Loader & Backhoe (90%)	10,440
Bulldozer	1,120
Dump	
Survey	1,335
Land	3,735
Total	$43,410

ANNUAL COST

Land	$ 460
Amortization	
Construction	1,435
Equipment	4,210
Operating Personnel	20,695
Repair and Maintenance	
Construction	265
Equipment	1,320
Energy	
Fuel	2,325
Electricity	230
Taxes	115
Insurance	435
Total	$31,490

Costs of Levels I, II and III treatment and disposal technologies are summarized in Table 23 for sludge residuals from primary lead smelting and refining. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric tons of lead product. Costs for each level of technology are also expressed as a percent of metal selling price.

Total industry costs for Levels I, II and III treatment and disposal technologies have been estimated in 1973 dollars and are presented in Table 23. The annualized industry costs for Level I and II treatment and disposal technologies is estimated as $180,000 which is 0.08% of estimated national sales. The annualized industry cost for Level III treatment and disposal technology (i.e. adequate for environmental protection) is estimated as $560,000 which is 0.25% of estimated 1973 national sales.

TABLE 23 COST SUMMARY FOR TREATMENT AND DISPOSAL TECHNOLOGIES: PRIMARY LEAD SMELTING AND REFINING

Annual Production: Model Plant-110,000 MT Industry-613,196 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.19

Cum. Unit Waste Disposal Costs:

	Level								
Waste (Type)	I			II			III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$2.06	$0.69	$0.39	$2.06	$0.69	$0.39	$2.90	$0.97	$0.55
Annual Cost	1.50	0.50	0.29	1.50	0.50	0.29	4.83	1.61	0.92
Total Capital Cost	-		$0.39	-		$0.39	-		$0.55
Total Annual Cost	-		0.29	-		0.29	-		0.92

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$0.24	$0.18	$0.24	$0.18	$0.34	$0.56
Total	$0.24	$0.18	$0.24	$0.18	$0.34	$0.56

1973 Metal Price: $358.38/MT

Percent Treatment Cost/Price of Metric Ton of Production

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.11%	0.08%	0.11%	0.08%	0.15%	0.26%
Total	0.11%	0.08%	0.11%	0.08%	0.15%	0.26%

3.0 PRIMARY SMELTING AND REFINING OF ZINC (SIC 3333)

3.1 INDUSTRY CHARACTERIZATION

There are two major processes employed for recovery of zinc metal from ore concentrates. They are electrolytic recovery and distillation (pyrometallurgical). Of the seven operating primary zinc plants in the United States three use electrolytic processes while four employ various modes of pyrometallurgical distillation (vertical retort, horizontal retort, electrothermic). Table 24 gives the location of primary zinc plants, types of operation, and zinc producing capacities. This table also indicates the presence or absence of byproduct acid plants and identifies major products produced at the plants. Table 25 gives state by state, EPA region and national primary lead production capacity by process type. Total U.S. production of refined primary zinc in 1973 was 628,785 short tons and the average selling price was 20.66 cents/lb (Ref. 2). National sales are therefore estimated as $259,814,000 in 1973.

The reduction of zinc ores and concentrates to zinc is accomplished by electrolytic deposition from a solution or by distillation in retorts or furnaces. For either method the zinc concentrate is roasted to eliminate most of the sulfur and for conversion to impure zinc oxide called roasted concentrates or "calcines".

At electrolytic zinc plants, the roasted zinc concentrate is leached with dilute sulfuric acid to form a zinc sulfate solution. The pregnant solution is then purified and piped to electrolytic cells, where the zinc is electrodeposited on aluminum cathodes. The cathodes are lifted from the tanks and are stripped of the zinc, which is then melted in a furnace and cast into slab form. Zinc concentrates shipped to electrolytic plants commonly contain lead, gold, and silver. The zinc acid leaching and electrolytic tank residues become enriched in these metals by the extraction of the zinc and are usually shipped to a lead smelter. There the lead, gold, and silver content is recovered in lead bullion.

Distillation retort plants are classified as horizontal retorts, vertical retorts that are externally heated by fuel, or continuous vertical retorts heated electrothermally. All employ coal or coke as the reducing agent. Quantities of coal or coke required range from about 0.5 to 0.8 ton per ton of slab zinc output. The zinc vapor and carbon monoxide from the retorts pass into condensers where the zinc is collected as liquid metal ready for casting into slab form. Zinc produced by distillation, normally the lower commercial grade, may be upgraded by refining to reduce the quantities of impurities. Refining, by redistillation is accomplished by vertical fractionating columns which separate the impurities contained in the feed zinc and can produce zinc of 99.995 plus purity.

Table 24

CHARACTERISTICS OF PRIMARY ZINC PLANTS IN THE UNITED STATES

COMPANY	LOCATION	TYPE OF OPERATION	ANNUAL ZINC PRODUCING CAPACITY MT/YR	ACID PLANT OPERATION	PRODUCTS PRODUCED AT PLANT SITE
AMERICAN SMELTING AND REFINING CO.	AMARILLO, TEXAS	HORIZONTAL RETORT	48,000	NO	PRIME WESTERN ZINC
AMERICAN SMELTING AND REFINING CO.	CORPUS CHRISTI, TEXAS	ELECTROLYTIC	98,000	YES	SLAB ZINC, ZINC ALLOYS, ZINC SULFATE, CADMIUM, SULFURIC ACID
AMAX	SAUGET, ILLINOIS	ELECTROLYTIC	76,000	YES	SLAB ZINC, CADMIUM, ZINC SULFATE, SULFURIC ACID
BUNKER HILL (GULF RESOURCES AND CHEMICAL CO.)	WALLACE, IDAHO	ELECTROLYTIC	96,000	YES	SLAB ZINC, ZINC ALLOYS, CADMIUM, SULFURIC ACID
NEW JERSEY ZINC (GULF AND WESTERN INDUSTRIES)	PALMERTON, PA.	VERTICAL RETORT	107,000	YES	SLAB ZINC, ZINC ALLOYS, ZINC DUST AND PELLETS, ZINC OXIDE, FERROALLOY, SULFURIC ACID
ST. JOE MINERALS CORP.	MONACA, PA.	ELECTROTHERMIC	227,000	YES	SLAB ZINC, ZINC ALLOYS, ZINC OXIDE, CADMIUM, FERROSILICON, MERCURY SULFURIC ACID
NATIONAL ZINC CO.	BARTLESVILLE, OKLA.	HORIZONTAL RETORT	57,000	YES	

SOURCE:
DRAFT EFFLUENT GUIDELINES
PRIMARY ZINC SMELTING AND
REFINING, EPA, JUNE, 1974

Table 25

GEOGRAPHICAL DISTRIBUTION OF PRIMARY ZINC SMELTERS AND REFINERS

STATE	NO. OF PLANTS	PLANT DISTRIBUTION BY CAPACITY (MT)			PLANT DISTRIBUTION BY PROCESS				CAPACITY PER PROCESS (MT)			
		50K-100K	100K-200K	200K-250K	ET	HR	VR	E	ET	HR	VR	E
IDAHO	1	1	0	0	0	0	0	1	0	0	0	96000
ILLINOIS	1	1	0	0	0	0	0	1	0	0	0	76000
OKLAHOMA	1	1	0	0	0	1	0	0	0	57000	0	0
PENNSYLVANIA	2	0	1	1	1	0	1	0	227000	0	107000	0
TEXAS	2	2	0	0	0	1	0	1	0	48000	0	98000
EPA REGION												
I	0	0	0	0	0	0	0	0	0	0	0	0
II	0	0	0	0	0	0	0	0	0	0	0	0
III	2	0	1	1	1	0	1	0	227000	0	107000	0
IV	0	0	0	0	0	0	0	0	0	0	0	0
V	1	1	0	0	0	0	0	1	0	0	0	76000
VI	3	3	0	0	0	2	0	1	0	105000	0	98000
VII	0	0	0	0	0	0	0	0	0	0	0	0
VIII	0	0	0	0	0	0	0	0	0	0	0	0
IX	0	0	0	0	0	0	0	0	0	0	0	0
X	1	1	0	0	0	0	0	1	0	0	0	96000
NATION	7	5	1	1	1	2	1	3	227000	105000	107000	270000

ET = ELECTROTHERMIC HR = HORIZONTAL RETORT VR = VERTICAL RETORT
E = ELECTROLYTIC

SOURCE: CALSPAN CORPORATION

3.2 WASTE CHARACTERIZATION

This section contains descriptions of production technology at primary zinc smelters and the resultant byproducts or wastes which are either recycled directly, shipped to other smelters for further metallic recovery, or disposed of on land. Estimates are given for the quantities of wastes and potentially hazardous constituents thereof which are disposed of on land either in lagoons or open dumps.

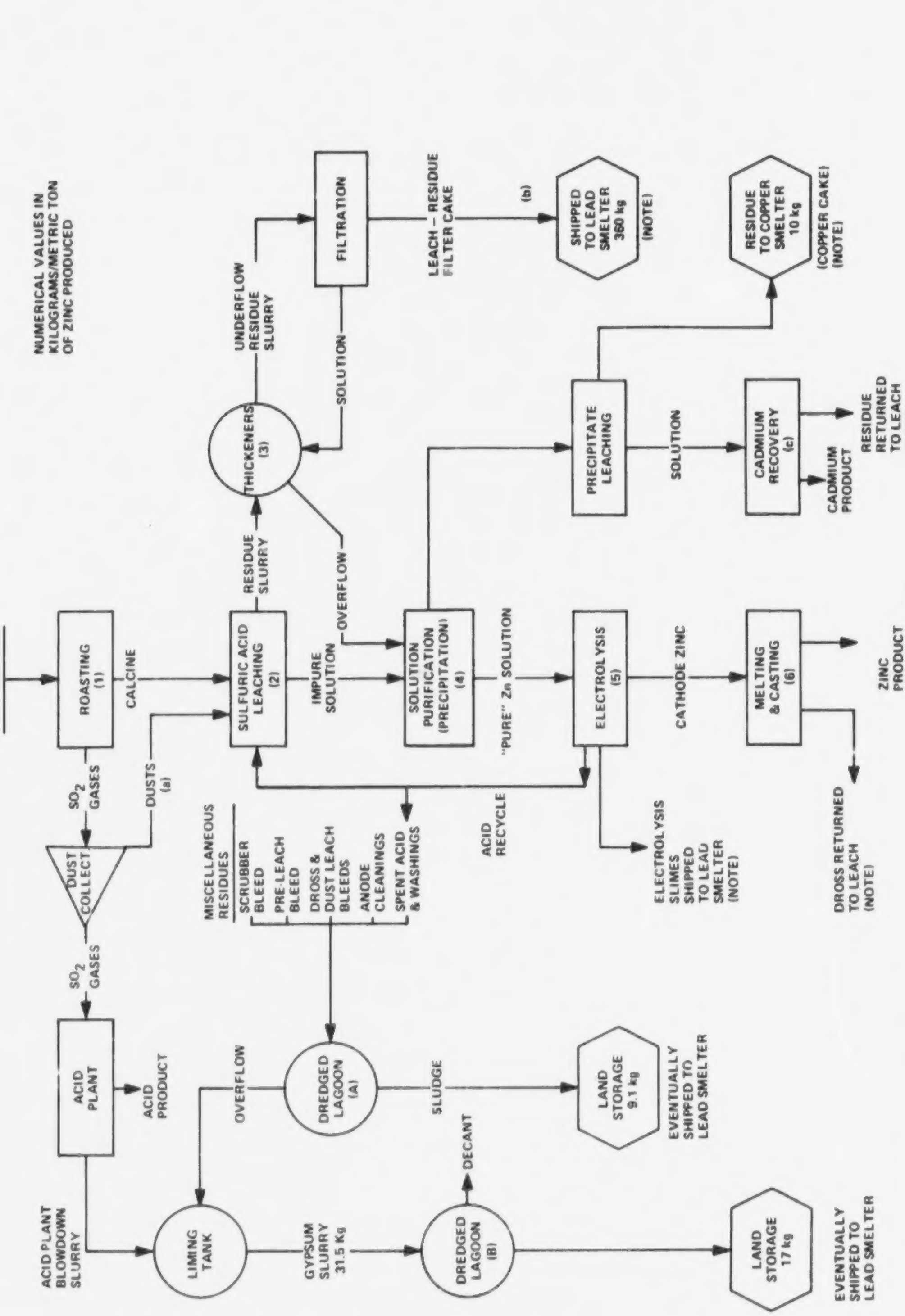
3.2.1 Process Description

Because of major differences in production processes and waste generation between electrolytic and pyrometallurgical (distillation) zinc plants, the two types of plants will be discussed separately.

Electrolytic. The hypothetical electrolytic zinc plant is assumed to have an annual capacity of 100,000 MT (110,000 short tons) per year. Two of the three operating electrolytic zinc plants in the U.S. are within ten percent of this size and the third is very nearly three-fourths of this size. Electrolytic zinc plants are assumed to operate 350 days/year; the daily capacity is, therefore, 286 MT (314 short tons/day).

Electrolytic reduction of zinc from sulfide ore concentrates begins with roasting (as does pyrometallurgical zinc smelting). However, the roasted concentrates are dissolved in sulfuric acid and processed in aqueous solution rather than being thermochemically treated to reduce and volatilize zinc. The electrolytic method is considered to be more efficient than the retort method in the recovery of lead, copper, and precious metal values associated with the zinc sulfide concentrates which constitute the primary feed to zinc reduction plants of both types.

A simplified flow diagram identifying solid waste sources and disposal is shown in Figure 4. The primary process steps in the electrolytic method comprise (1) roasting the zinc sulfide concentrate to reduce the sulfur content, (2) leaching the calcine with acidic stripping electrolyte, (3) thickening and filtration to remove the leach residue, (4) oxidation of the dissolved iron followed by neutralization of acid with lime, calcine, and metallic zinc to precipitate copper, iron, arsenic, antimony, cadmium, germanium, nickel, and cobalt, (5) electrolysis of the purified zinc and an acidic stripped electrolyte for reuse in leaching fresh calcine. The cathode deposit of zinc is melted and casted into high-purity slab zinc. (6) The auxiliary operations include (a) collecting and returning all dusts and fumes to the roasters, (b) washing and drying the zinc leach residue with its lead and precious metal content for reprocessing at a lead smelter, (c) selective leaching of the solution purification residue to recover a cadmium-rich solution for electrolysis with zinc as a returnable solution, and copper as a


NUMERICAL VALUES IN KILOGRAMS/METRIC TON OF ZINC PRODUCED
ZINC SULFIDE CONCENTRATES
ROASTING (1)
SO2 GASES
DUST COLLECT.
DUSTS (a)
CALCINE
SULFURIC ACID LEACHING (2)
RESIDUE SLURRY
THICKENERS (3)
UNDERFLOW RESIDUE SLURRY
FILTRATION
SOLUTION
LEACH – RESIDUE FILTER CAKE
(b)
SHIPPED TO LEAD SMELTER 360 kg
(NOTE)
OVERFLOW
IMPURE SOLUTION
SOLUTION PURIFICATION (PRECIPITATION) (4)
PRECIPITATE LEACHING
RESIDUE TO COPPER SMELTER 10 kg
(COPPER CAKE) (NOTE)
SOLUTION
CADMIUM RECOVERY (c)
CADMIUM PRODUCT
RESIDUE RETURNED TO LEACH (NOTE)
"PURE" Zn SOLUTION
ELECTROLYSIS (5)
CATHODE ZINC
MELTING & CASTING (6)
ZINC PRODUCT 1.0 METRIC TON
DROSS RETURNED TO LEACH (NOTE)
ACID RECYCLE
ELECTROLYSIS SLIMES SHIPPED TO LEAD SMELTER (NOTE)
MISCELLANEOUS RESIDUES
SCRUBBER BLEED
PRE-LEACH BLEED
DROSS & DUST LEACH BLEEDS
ANODE CLEANINGS
SPENT ACID & WASHINGS
SO2 GASES
ACID PLANT
ACID PRODUCT
ACID PLANT BLOWDOWN SLURRY
LIMING TANK
OVERFLOW
DREDGED LAGOON (A)
SLUDGE
LAND STORAGE 9.1 kg
EVENTUALLY SHIPPED TO LEAD SMELTER
GYPSUM SLURRY 31.5 Kg
DREDGED LAGOON (B)
DECANT
LAND STORAGE 17 kg
EVENTUALLY SHIPPED TO LEAD SMELTER

NOTE – NOT STORED OR DRIED ON LAND

Figure 4 PRIMARY ZINC REFINING (ELECTROLYTIC)

residue for shipment to a copper smelter. In addition to these operations, part of the spent electrolyte must periodically or continuously be bled from the circuit for periodic purification or discharge to waste to remove impurities, such as chlorine, fluorine, cobalt, magnesium, and sodium that gradually build up in the electrolyte.

Pyrometallurgical. The hypothetical pyrometallurgical zinc plant is assumed to have an annual capacity of 107,000 MT (118,000 short tons) of 99.+% zinc metal. This figure is based on the approximate size of the smaller of the two actual vertical retort plants operating in the U.S. and on the fact that the larger of the two operating plants has approximately double this capacity at 227,000 MT (250,000 short tons). No consideration is given to the size of the two remaining horizontal retort zinc plants in the U.S. because both are scheduled to cease operations.

Pyrometallurgical zinc smelters are estimated to operate 350 days/year; the daily production and assumed capacity of the hypothetical plant is, therefore, 310 MT (338 short tons).

Input (feed) to the plant consists of roughly 214,000 MT (236,000 short tons) annually of sulfide concentrates, oxide ores, and zinc scrap averaging roughly 50% zinc in the aggregate.

The major operational steps in the vertical retort process are (1) Roasting to decrease the sulfur content of the zinc concentrate; (2) sintering to remove residual sulfur and volatilize cadmium, lead, indium, germanium and selenium; (3) retorting a thoroughly mixed charge of sinter, and carbon reductant in either a vertical retort or electrothermic furnace; and (4) condensing the volatilized zinc. Roaster gases are scrubbed free of dust, and the sulfur dioxide content converted to sulfuric acid (5). Dust and fumes from the roasters and sinter hearths are leached with sulfuric acid to recover cadmium (6). Oxide ores, scrap and reverts are pretreated in a Waelz kiln to upgrade the zinc content (7).

Figure 5 is a smiplified flow sheet showing the sources of solid wastes and their disposal. The main retort residue is composed of residual coked coal, residual zinc, ferrosilicon, and a siliceous slag component.

3.2.2 Description of Waste Streams

Electrolytic. a. The chief solid residue in the electrolytic reduction of zinc from sulfide ore concentrates is the leach residue (i.e., that portion of the calcined feed which fails to dissolve in the sulfuric acid leach solution). This residue does not, however, constitute a waste at the typical installation; it is shipped immediately to a lead smelter for recovery of lead and precious metal values.

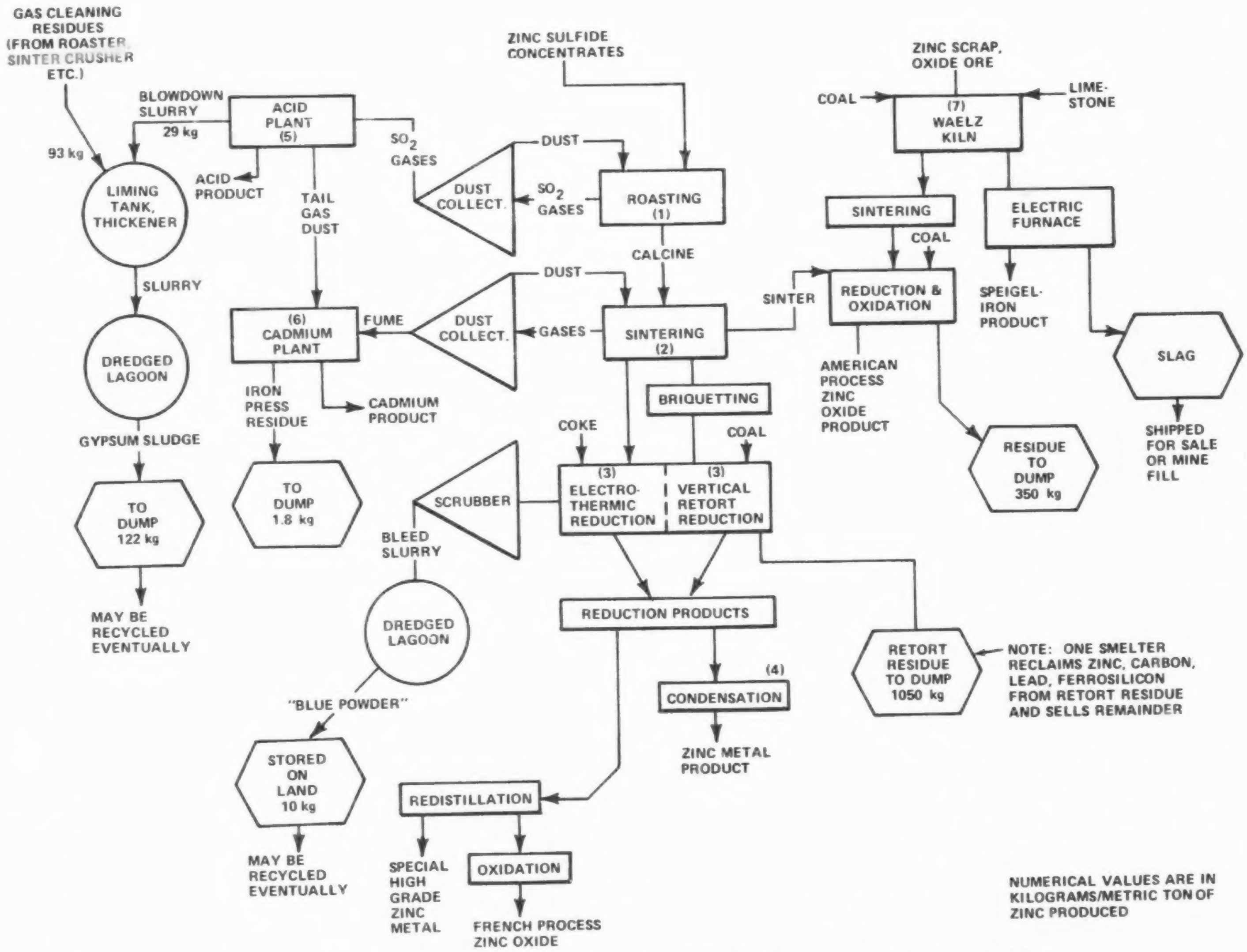

GAS CLEANING RESIDUES (FROM ROASTER, SINTER CRUSHER ETC.)
93 kg
BLOWDOWN SLURRY
29 kg
ACID PLANT (5)
ACID PRODUCT
TAIL GAS DUST
SO_2 GASES
LIMING TANK, THICKENER
SLURRY
DREDGED LAGOON
GYPSUM SLUDGE
TO DUMP 122 kg
MAY BE RECYCLED EVENTUALLY
DUST COLLECT.
DUST
SO_2 GASES
ZINC SULFIDE CONCENTRATES
ROASTING (1)
CALCINE
(6) CADMIUM PLANT
FUME
DUST COLLECT.
GASES
SINTERING (2)
IRON PRESS RESIDUE
CADMIUM PRODUCT
TO DUMP 1.8 kg
BRIQUETTING
COKE
COAL
(3) ELECTRO-THERMIC REDUCTION
(3) VERTICAL RETORT REDUCTION
SCRUBBER
BLEED SLURRY
DREDGED LAGOON
"BLUE POWDER"
STORED ON LAND 10 kg
MAY BE RECYCLED EVENTUALLY
REDUCTION PRODUCTS
(4)
CONDENSATION
ZINC METAL PRODUCT
REDISTILLATION
SPECIAL HIGH GRADE ZINC METAL
OXIDATION
FRENCH PROCESS ZINC OXIDE
ZINC SCRAP, OXIDE ORE
COAL
LIME-STONE
(7) WAELZ KILN
SINTERING
COAL
ELECTRIC FURNACE
SINTER
REDUCTION & OXIDATION
AMERICAN PROCESS ZINC OXIDE PRODUCT
SPEIGEL-IRON PRODUCT
SLAG
SHIPPED FOR SALE OR MINE FILL
RESIDUE TO DUMP 350 kg
RETORT RESIDUE TO DUMP 1050 kg
NOTE: ONE SMELTER RECLAIMS ZINC, CARBON, LEAD, FERROSILICON FROM RETORT RESIDUE AND SELLS REMAINDER
NUMERICAL VALUES ARE IN KILOGRAMS/METRIC TON OF ZINC PRODUCED

Figure 5 PRIMARY ZINC SMELTING AND REFINING (PYROMETALLURGICAL)

b. Miscellaneous Acidic Slurries. Acidic residue from sources such as gas scrubber bleed, preleach bleed, dross and dust leach bleed, anode washing, and spent acid bleed contain suspended and dissolved solids, as well as sulfuric acid and heavy metals (Cd, Cu, Pb, Zn). These water slurries in the amount of 1,480 cu m/day (391,700 gal/day) are settled in an unlined lagoon (A) of capacity 2000 cu m (72,000 cu ft.). Settled solids are dredged from the lagoon and dried on land near the plant prior to eventual shipment to a lead smelter. The amount of sludge dredged (on a dry basis) is 9.1 Kg/MT of zinc product or 2.6 MT/day. Wet sludge generation will be in the order of 23 MT/day.

Overflow from the above lagoon goes to a lime treatment facility where it joins with acid plant blowdown slurry as discussed below.

c. Acid Plant Blowdown Slurry. The amount of calcium sulfate- sulfite sludge resulting from lime treatment of acid plant blowdown as dry weight is approximately 31.5 Kg/MT zinc product (approximately 105 Kg/MT wet weight). Table 26 contains estimates of the daily flow volumes and solids contents of limed slurry from acid plant blowdown plus limed overflow from the lagoon (A) in which the above-described miscellaneous acid slurries are settled. These solids are dredged from a second unlined lagoon (B).

Solids dredged from lagoon B (17 Kg/MT of zinc product or 4.8 MT/day) are dried on land for eventual shipment to a lead smelter.

TABLE 26

SOLIDS IN TWO LIMED EFFLUENT STREAMS PRESENT AT AN ELECTROLYTIC ZINC PLANT

Stream	Flow cu m/day	Flow gal/day	Solids (Dredged from Lagoon B) MT/day	Solids (Dredged from Lagoon B) S. ton/day	Solids (Dredged from Lagoon B) MT/year
Lagoon (A) Overflow	1,480	392,000	8.5	9.4	2,960
Acid Plant Blowdown (B)	1,380	364,000	9.0	9.9	3,150
Combined	2,860	756,000	17.5	19.3	6,110

d. Other Residues. As shown in Figure 4 other residues yielded by the component processes of electrolytic zinc reduction are recycled immediately or shipped to copper or lead smelters. Since these materials are not stored or dried on the ground, they are not considered to be a part of the solid waste disposal scenario.

Pyrometallurgical

a. Retort Furance Residue. The chief solid waste from the hypothetical zinc smelter is a retort residue amounting to 1050 Kg/MT of zinc product or 322 MT/day (353 short tons/day). This material is transported to the main dump by rail car. At one of the two operating vertical retort plants, the retort residue is processed to recover coke, unsmelted zinc, low-grade ferrosilicon, and slag. The zinc-lean slag is sized and sold as road ballast.

b. Slurries from Primary Gas Cleaning Operations. Roaster gases laden with SO_2 are partially scrubbed of particulates with baghouses or electrostatic precipitators before entering the acid plant. Gases from other primary operations, such as sintering and roasting and crushing are similarly cleaned in electrostatic precipitators, wet scrubbers, and baghouses. Most of the dusts and slurries resulting from these primary gas cleaning operations are recycled immediately, but a portion of the slurries amounting to 90 Kg dry sludge/MT of zinc product (225 Kg/MT wet weight) is limed and thickened (in conjunction with acid plant blowdown slurry, see below). Underflow from the thickener is partially recycled, but the overflow (and some underflow) goes to an unlined settling lagoon. Flows and solids quantities are tabulated below along with those attributable to acid plant blowdown.

c. Acid Plant Blowdown Slurry. The amount of calcium sulfate-sulfite sludge resulting from lime treatment of acid plant blowdown as dry weight is 31 Kg/MT zinc product (approximately 100 Kg/MT wet weight). Table 27 contains estimates of the daily flow volumes and solids contents of limed slurries from acid plant blowdown and primary gas cleaning operations which are combined prior to liming and then settled as an intimate mixture in the lagoon before dredging.

TABLE 27

SOLIDS IN TWO LIMED EFFLUENT STREAMS PRESENT AT A
PYROMETALLURGICAL ZINC PLANT

	Flow		Solids (Dredged from Lagoon)		
Stream	cu m/day	gal/day	MT/day	Short ton/day	MT/year
Primary Gas Cleaning Operation (Sinter, Roasters, Crushers)	1,090	288,000	28	31	9,950
Acid Plant Blowdown	1,310	345,000	9.0	9.9	3,150
Combined	2,400	633,000	37	41	13,100

The total amount of dredged sludges from these sources amounts to 122 Kg/MT of zinc product or 37 MT/day or 13,100 MT/year (37,430 MT/year wet weight). The dredged sludges are either stored for long periods of time (i.e. months) before recycling or disposed permanently on land.

d. Retort Gas Scrubber Bleed Slurry ("Blue Powder"). A small portion of the zinc product vapor produced in the retorts passes through the zinc condensers and is recovered as "blue powder" from wet scrubber bleed slurry and settled in two concrete pits. Slurry volume flowing into the concrete pits is 136 to 272 cu m/day (1500 to 3000 gal/day). Each pit is approximately 20' x 40' in size. It is estimated that 10 Kg/MT of zinc product or 3.1 MT/day (3.4 short tons/day) of blue powder is dredged from the pits and that the amount stored on the ground for months for drying and oxidation before eventual recycle is roughly 535 MT at any given time.

e. Cadmium Plant Residue. Some fume and dust from primary gas cleaning operations is treated in a hydrometallurgical plant to recover cadmium metal. A filter cake residue amount to 1,8 Kg/MT of zinc product or 0.514 MT/day (0.566 short tons/day) is trucked to the main retort residue dump for disposal.

Chemical analyses of samples obtained from primary zinc plants are given in Appendix A.

Retort residue consists of friable 3-5" diameter gravel like lumps. In solubility tests as described in Appendix B the only heavy metal found to leach significantly was zinc (230 ppm in leachate). Zinc is not a highly toxic metal and a plant and animal trace nutrient. Retort residue, in view of available information, is not considered potentially hazardous at this time.

Cadmium plant residue contains high concentrations of cadmium, copper, lead and zinc. Lead was found to leach to the extent of 9 ppm and zinc to the extent of 4,000 ppm from cadmium plant residue. This residue is therefore considered potentially hazardous.

Sludges from gas cleaning, acid plant blowdown lime treatment (i.e. gypsum sludges) and anode sludges contain high concentrations of the highly toxic heavy metals cadmium and lead and high concentrations of zinc and copper also. These elements were found to leach significantly in solubility tests described in Appendix A. The silt to colloidal sizes of individual sludge particulates make sludges susceptible to weathering processes. For the above reasons sludges are considered potentially hazardous when disposed or stored on land for periods of months or greater.

3.2.3 Waste Quantities

Tables 28 and 29 summarize the generation factors for each of the land disposed or stored residuals in the electrolytic and pyrometallurgical primary zinc smelting and refining categories. These tables also gives typical concentrations of potentially hazardous constituents. Table 30 gives the yearly quantity of wastes and potentially hazardous constituents for a typical plant producing 100,000 MT (110,000 short tons) of zinc metal by the electrolytic process. Table 31 gives this information for a typical plant producing 107.000 MT (118,000 short tons) of zinc metal per year using pyrometallurgical processes (i.e. vertical retort or electrothermic).

Using the waste generation factors and concentration factors contained in Tables 28 and 29 and smelter capacities estimates have been made for the quantities of land disposed or stored residuals and hazardous constituents thereof. These estimates are given for 1974, 1977 and 1983 on a state- by-state, regional and national level in Tables 32 and 33.

It will be noted that the amount of potentially hazardous sludge from pyrometallurgical zinc production is expected to decrease over the period 1974-1983 while the amount of potentially hazardous sludge from electrolytic refining is expected to increase significantly, especially from 1977 to 1983. This is because pyrometallurgical zinc capacity is expected to decrease over this period while electrolytic capacity is expected to increase. The net result of a technology shift to electrolytic refining of zinc from pyrometallurgical refining should be a marked decrease in sludge generation since pyrometallurgical operations produce approximately 125 Kg of sludge/MT of zinc product and electrolytic produces approximately 25 Kg/MT of product.

Projections of land disposed waste for 1977 are based on increased electrolytic plant capacity now under construction in Oklahoma (51,000 MT). Further planned increases in electrolytic plant capacity in Kentucky (163,010 MT) and Tennessee (145,000 MT) (Reference 9) account for estimated waste increases by 1983.

On the other hand the closing of pyrometallurgical plants in Oklahoma and Texas will result in no wastes from pyrometallurgical plants in these states in 1977 and 1983. Pyrometallurgical plant expansion in Pennsylvania will increase land disposed waste by an estimated 11 percent.

3.3 TREATMENT AND DISPOSAL TECHNOLOGY

3.3.1 Current Waste Treatment and Disposal Practices

For purposes of clarity, treatment and disposal of residues will be discussed separately for electrolytic and pyrometallurgical plants.

TABLE 28 WASTE RESIDUAL FACTORS, PRIMARY ZINC ELECTROLYTIC REFINING

Type of Waste	Generation Factor kg/MT	Concentration Factor (ppm)							
		Cd	Cr	Cu	Hg	Mn	Pb	Se	Zn
Sludges	26.1	820	44	2510	22	8740	15,300	66	220,000

TABLE 29 WASTE RESIDUAL FACTORS, PRIMARY ZINC SMELTING AND REFINING (PYROMETALLURGICAL)

Type of Waste	Generation Factor kg/MT	Concentration Factor (ppm)						
		Cd	Cr	Cu	Pb	Se	Zn	Hg
Retort Residue	1050	850	45	4,630	2,410	8.3	106,900	-
Sludges	122	822	31	540	2,920	46	306,900	19

TABLE 30 YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL PLANT, PRIMARY ZINC ELECTROLYTIC REFINING

Type of Waste	Total Waste Quantity (MT)	Quantity of Potentially Hazardous Constituents (MT)							
		Cd	Cr	Cu	Hg	Mn	Pb	Se	Zn
Sludges	2610	2.1	0.11	6.6	0.06	23	40	0.17	575

*Quantities calculated from generation and concentration factors given in Table 28 based on annual production of 100,000 MT. Divide by 350 to obtain daily quantities, multiply by 1.1 to convert to short tons.

TABLE 31 YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL PLANT, PRIMARY ZINC SMELTING & REFINING (PYROMETALLURGICAL)

Type of WAste	Total Waste Quantity, (MT)	Quantity of Potentially Hazardous Constituents (MT)						
		Cd	Cr	Cu	Pb	Se	Zn	Hg
Retort Residue	112,400	95.8	5.1	520	271	0.93	12,000	-
Sludges	13,050	10.7	0.40	7.0	38	0.60	3,998	0.25

* Quantities calculated from generation and concentration factors given in Table 29 based on annual production of 107,000 MT. Divide by 350 to obtain daily quantities, multiply by 1.1 to convert to short tons.

Table 32-a
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ELECTROLYTIC ZINC SMELTING AND REFINING TOTAL SLUDGES - 1974 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS							
					Cd	Cr	Cu	Hg	Mn	Pb	Se	Zn
ILLINOIS	487	487 (A)	210	LAGOON	0.01	0.01	0.03		143	60	0.03	8.5
IDAHO	8,840	8,840 (A)	2,200	LAGOON	12.3	0.4	23	0.2	9.5	110	0.6	2000
TEXAS	8,960	8,960 (A)	2,200	LAGOON	12.3	0.4	23	0.2	9.5	110	0.6	2000
EPA REGION												
V	487	487	210		0.01	0.01	0.03		143	60	0.03	8.5
VI	8,960	8,960	2,200		12.3	0.4	23	0.2	9.5	110	0.6	2000
X	8,840	8,840	2,200		12.3	0.4	23	0.2	9.5	110	0.6	2000
NATIONAL TOTAL	18,300	18,300 (A)	4,600		15	0.8	46	0.4	160	280	1.2	4010

NOTES:

(A) SOME PERCENTAGE OF THESE SLUDGES IS DREDGED FROM LAGOONS, DRIED AND STORED ON LAND, AND EVENTUALLY RECYCLED. THE REMAINDER IS PERMANENTLY LAND-DISPOSED IN OPEN DUMPS OR LARGE TAILINGS PONDS.

*INCLUDES ACID PLANT GYPSUM SLUDGE PLUS SLUDGES FROM SCRUBBERS, ANODES, BLEEDS, SPENT ACIDS AND WASHINGS. MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS. MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 32-b

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ELECTROLYTIC ZINC SMELTING AND REFINING TOTAL SLUDGES - 1977 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS							
					Cd	Cr	Cu	Hg	Mn	Pb	Se	Zn
ILLINOIS	487	487(A)	210	LAGOON	0.01	0.01	0.03		143	60	0.03	8.5
IDAHO	8,840	8,840(A)	2,200	LAGOON	12.3	0.4	23	0.2	9.5	110	0.6	2000
OKLAHOMA	4,659	4,659	1,144		6.4	0.2	12	0.1	4.9	57	0.3	1040
TEXAS	8,960	8,960(A)	2,200	LAGOON	12.3	0.4	23	0.2	9.5	110	0.6	2000
EPA REGION												
V	487	487	210		0.01	0.01	0.03		143	60	0.03	8.5
VI	13,619	13,619	3,344		18.7	0.6	35	0.3	14.4	167	0.9	3040
X	8,840	8,840	2,200		12.3	0.4	23	0.2	9.5	110	0.6	2000
NATIONAL TOTAL	22,946	22,946(A)	5,754		31.0	1.0	58.0	0.5	166.9	337	1.5	5048

NOTES:

(A) SOME PERCENTAGE OF THESE SLUDGES IS DREDGED FROM LAGOONS, DRIED AND STORED ON LAND, AND EVENTUALLY RECYCLED. THE REMAINDER IS PERMANENTLY LAND-DISPOSED IN OPEN DUMPS OR LARGE TAILINGS PONDS.

*INCLUDES ACID PLANT GYPSUM SLUDGE PLUS SLUDGES FROM SCRUBBERS, ANODES, BLEEDS, SPENT ACIDS AND WASHINGS. MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS. MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 32-c

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ELECTROLYTIC ZINC SMELTING AND REFINING TOTAL SLUDGES - 1983 (METRIC TONS, DRY WEIGHTS) *

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS							
					Cd	Cr	Cu	Hg	Mn	Pb	Se	Zn
ILLINOIS	487	487(A)	210	LAGOON	0.01	0.01	0.03		143	60	0.03	8.5
IDAHO	8,840	8,840(A)	2,200	LAGOON	12.3	0.4	23	0.2	9.5	110	0.6	2000
KENTUCKY	14,874	14,874	3,652		20.4	0.7	38	0.3	15.8	183	1.0	3320
OKLAHOMA	4,659	4,659	1,144		6.4	0.2	12	0.1	4.9	57	0.3	1040
TENNESSEE	13,261	13,261	3,256		18.2	0.6	34	0.3	14.1	163	0.9	2960
TEXAS	8,960	8,960(A)	2,200	LAGOON	12.3	0.4	23	0.2	9.5	110	0.6	2000
EPA REGION												
IV	28,135	28,135	6,908		38.6	1.3	72	0.6	29.9	346	1.9	6280
V	487	487	210		0.01	0.01	0.03		143	60	0.03	8.5
VI	13,619	13,619	3,344		18.7	0.6	35	0.3	14.4	167	0.9	3040
X	8,840	8,840	2,200		12.3	0.4	23	0.2	9.5	110	0.6	2000
NATIONAL TOTAL	51,081	51,081(A)	12,662		69.6	2.3	130	1.1	196.8	683	3.4	11328

NOTES:

(A) SOME PERCENTAGE OF THESE SLUDGES IS DREDGED FROM LAGOONS, DRIED AND STORED ON LAND, AND EVENTUALLY RECYCLED. THE REMAINDER IS PERMANENTLY LAND-DISPOSED IN OPEN DUMPS OR LARGE TAILINGS PONDS.

*INCLUDES ACID PLANT GYPSUM SLUDGE PLUS SLUDGES FROM SCRUBBERS, ANODES, BLEEDS, SPENT ACIDS AND WASHINGS.
MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 33-a
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY PYROMETALLURGICAL ZINC SMELTING AND REFINING TOTAL SLUDGES - 1974 (METRIC TONS, DRY WEIGHT)*

STATE	TOTAL DISPOSAL	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS						
					Cd	Cr	Cu	Hg	Pb	Se	Zn
PENNSYLVANIA	58,000	58,000 (A)	18,804	LAGOON	49	1.7	30	1.0	120	2.5	18,600
TEXAS	6,300	6,300 (A)	1,611	LAGOON	4.6	0.2	4.2	0.2	42	0.4	1,560
OKLAHOMA	7,500	7,500 (A)	1,921	LAGOON	5.5	0.3	4.9	0.2	50	0.4	1,860
EPA REGION											
III	58,000	58,000	18,804		49	1.7	30	1.0	120	2.5	18,600
VI	13,800	13,800	3,533		10	0.5	9.1	0.4	92	0.8	3,420
NATIONAL TOTAL	71,800	71,800 (B)	22,336		59	2.2	39	1.4	210	3.3	22,000

NOTES:

(A) A LOW PERCENTAGE OF THESE SLUDGES MAY BE PERMANENTLY DISPOSED ON A DUMP AFTER DREDGING FROM A LAGOON OR MAY BE PERMANENTLY LEFT IN LARGE LAGOONS OR TAILINGS PONDS.

(B) A LARGE PORTION OF THIS MATERIAL IS RECYCLED AFTER DREDGING AND STORAGE ON LAND.

*INCLUDES ACID PLANT GYPSUM SLUDGE PLUS RESIDUES FROM CLEANING OF EMISSIONS FROM ROASTER, SINTER AND CRUSHER.

MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 33-b

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY PYROMETALLURGICAL ZINC SMELTING AND REFINING TOTAL SLUDGES - 1977, 1983 (METRIC TONS, DRY WEIGHT)*

STATE	TOTAL DISPOSAL	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS						
					Cd	Cr	Cu	Hg	Pb	Se	Zn
PENNSYLVANIA	64,380	64,380 (B)	20,872	LAGOON	54	1.9	33	1.1	133	2.8	20,646
EPA REGION III	64,380	64,380	20,872		54	1.9	33	1.1	133	2.8	20,646
NATIONAL TOTAL	64,380	64,380 (A)	20,872		54	1.9	33	1.1	133	2.8	20,646

NOTES:

(A) A LARGE PORTION OF THIS MATERIAL IS RECYCLED AFTER DREDGING AND STORAGE ON LAND.

(B) A LOW PERCENTAGE OF THESE SLUDGES MAY BE PERMANENTLY DISPOSED ON A DUMP AFTER DREDGING FROM A LAGOON OR MAY BE PERMANENTLY LEFT IN LARGE LAGOONS OR TAILINGS PONDS.

* INCLUDES ACID PLANT GYPSUM SLUDGE PLUS RESIDUES FROM CLEANING OF EMISSIONS FROM ROASTER, SINTER AND CRUSHER.

MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 33-c
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY PYROMETALLURGICAL ZINC SMELTING AND REFINING TOTAL RETORT RESIDUE, 1974 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	DISPOSAL METHOD	CONSTITUENTS					
				Cd	Cr	Cu	Pb	Se	Zn
PENNSYLVANIA	109,000	0	OPEN DUMP	93	4.9	500	260	0.9	11,700
TEXAS	48,900		OPEN DUMP	42	2.2	230	120	0.4	5,200
OKLAHOMA	58,000		OPEN DUMP	49	2.6	270	140	0.5	6,200
EPA REGION									
III	109,000			93	4.9	500	260	0.9	11,700
VI	107,000			91	4.8	500	260	0.9	11,400
NATIONAL TOTAL	216,000 (A)	↓		184	9.7	1000	520	1.8	23,100

NOTES:

(A) A SMALL PORTION OF THIS MATERIAL IS SOLD FOR CONSTRUCTION USES.

*THIS WASTE BELIEVED NONHAZARDOUS BASED ON CALSPAN SOLUBILITY TESTS DESCRIBED IN APPENDIX B.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 33-d
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY PYROMETALLURGICAL ZINC SMELTING AND REFINING TOTAL RETORT RESIDUE - 1977, 1983 (METRIC TONS) *

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	DISPOSAL METHOD	CONSTITUENTS					
				Cd	Cr	Cu	Pb	Se	Zn
PENNSYLVANIA	120,990	0	OPEN DUMP	103	5.4	555	289	1.0	12,987
EPA REGION III	120,990	↓		103	5.4	555	289	1.0	12,987
NATIONAL TOTAL	120,990(A)	↓		103	5.4	555	289	1.0	12,987

NOTES:

(A) A SMALL PORTION OF THIS MATERIAL IS SOLD FOR CONSTRUCTION USES.

*THIS WASTE BELIEVED NONHAZARDOUS BASED ON CALSPAN SOLUBILITY TESTS DESCRIBED IN APPENDIX B.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Electrolytic. Acid plant gypsum sludge and sludge from scrubber bleeds, anode cleaning, spent acid and other bleeds is currently dredged from unlined lagoons and either disposed of or stored on the ground for variable periods of time (i.e. a few too many months) before shipment to lead smelters for recovery of lead and other metals. These practices are inadequate because of the danger of heavy metal leaching as discussed previously.

Pyrometallurgical. Currently all but one pyrometallurgical plant disposes of retort residue and cadmium plant residue (iron press residue) by open dumping. This practice is considered adequate at the present time for retort residue but not cadmium plant residue which may be leached of heavy metals.

Gypsum sludge from the acid plant and blowdown bleed slurry (i.e. "blue powder") from the retorts are kept temporarily in unlined lagoons which are periodically dredged. Dredged sludges are stored on land and may eventually be recycled. As discussed previously these sludges contain potentially hazardous heavy metals including arsenic, lead, cadmium, zinc and copper and released some of these constituents in solubility tests.

The following sections describe levels of technology for those residuals from electrolytic or pyrometallurgical smelting and refining which are considered potentially hazardous.

3.3.2 Present Treatment and Disposal Technology (Level I)

Electrolytic. Settling of sludges and bleeds (i.e. acid plant gypsum sludge, scrubber bleed, anode cleaning spent acid) in unlined lagoons with disposal or storage on unsealed soil areas is environmentally inadequate if heavy metals leach through permeable soils to ground water.

Pyrometallurgical. Open dumping of cadmium plant residue is environmentally inadequate because of the possibility of heavy metal leaching. Temporary storage of acid plant gypsum sludge and blowdown bleed slurry (i.e. "blue powder") in unlined lagoons and storage of dredged sludges on land for considerable time periods (one month to many months) is environmentally inadequate because of the threat of heavy metal leaching and percolation through permeable soils to groundwater.

3.3.3 Best Technology Currently Employed (Level II)

Electrolytic. The technology employed for the settling and/or storage of sludges and bleeds is the same as that employed for Level I.

Pyrometallurgical. At least one plant is known to ship iron press residue from the cadmium plant to a lead smelter for further metal recovery. Immediate shipment of this residue will comprise Level II technology and is environmentally adequate.

Dredged scrubwater sediments from retorts ("blue powder") is recycled at one plant without ground storage and is environmentally adequate.

Storage or disposal of acid plant sludge on unsealed ground areas is Level II technology and is environmentally inadequate.

3.3.4 Technology To Provide Adequate Health and Environmental Protection (Level III)

Electrolytic. Lined lagoons and concrete storage pits are needed to prevent percolation of leached toxic heavy metals through permeable soils to groundwater. Alternatively, dredged sludges could be shipped immediately to lead smelters without ground storage.

Pyrometallurgical. Immediate shipment of iron press residue from the cadmium plant and immediate recycle of retort scrubwater sediments ("blue powder") will preclude environmental contamination.

Lined lagoons are needed for settling of sludge from primary gas cleaning and the acid plant. Lagoon sludge must be chemically fixed before dumping on land to prevent leaching of heavy metals and percolation through permeable soils to groundwater.

Tables 34, 35 and 36 summarize features of Levels I, II and III treatment and disposal technologies for electrolytic and pyrometallurgical plants including assessment of the hazardous nature of residuals and the methods and adequacy of treatment and disposal methods.

3.4 COST ANALYSIS

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. Costs are given separately for electrolytic plants and pyrometallurgical plants. The hypothetical electrolytic zinc plant is assumed to have an annual capacity of 100,000 MT of refined slab zinc while the capacity of the pyrometallurgical plant is assumed as 107,000 MT per year. Both plants operate 350 days per year.

Table 34

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary (Electrolytic) Zinc SIC 3333

SLUDGES (Acid plant, anode washings, bleeds, spent acid)

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Acid Plant - aggregated colloidal and silt size particulates; predominantly $CaSO_3$, $CaSO_4$	Same as I	Same as I
	Misc. - colloidal to sand size particulates, oxides, silicates	Same as I	Same as I
Amount of Waste (kg/MT Product)	Acid Plant -17 kg Misc. - 9.1 kg (Anode washings, Bleeds, Spent Acid)	Same as I	Same as I
Factors Affecting Hazardousness	Contain heavy metals including Cd, Cr, Zn, Cu, Pb, As	Same as I	Same as I
Treatment/ Disposal Technology	Unlined lagoons - dredged sludge stored on ground before shipment to lead smelters or disposed on ground	Same as I	Lined lagoons and storage of dredged sludge in concrete pits, or immediate shipment to lead smelters
Estimate of # + % of Plants Using Technology	3 (100%)	3 (100%)	0

Table 34-(cont'd.)

SLUDGES - Continued

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if heavy metals are leached	Same as I	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	Possible contamination of ground or surface water from heavy metals	Same as I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	Same as I	Groundwater and runoff water monitoring
Energy Requirements	Negligible	Negligible	Negligible

Table 35

Treatment and Disposal Technology Levels

Smelting and Refining Category PRIMARY PYROMETALLURGICAL ZINC SIC 3333

IRON PRESS RESIDUE FROM CADMIUM PLANT

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Iron Press Residue filter cake of silt size particles	No waste	same as II
Amount of Waste (kg/MT Product)	Iron press residue - 1.8 kg	No waste	No waste
Factors Affecting Hazardousness	Contain heavy metals including As, Cu, Pb, Zn, Cd, Cr, Se	No hazard	No hazard
Treatment/ Disposal Technology	Open Dumping	Iron press residue-immediate shipment to lead smelter	Iron press residue immediate shipment to lead smelter
Estimate of # + % of Plants Using Technology	3 (75%)	1 (25%)	Same as II

Table 35 - (cont'd.)

IRON PRESS RESIDUE -(Continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate	Adequate	Adequate
Problems and Comments	Solubility tests indicate leaching of zinc and lead from cadmium plant residues ("iron press")	—	—
Non-Land Environmental Impact	Possible contamination of ground or surface water if leached	None	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	Surveillance to assure reprocessing of iron press residues	Same as II
Energy Requirements	Negligible	Negligible	Negligible

Table 36

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary Zinc Pyrometallurgical SIC 3333

SLUDGES (acid plants retort scrubber)

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Environmental Protection)
Physical + Chemical Properties	Acid Plant sludge-aggregated colloidal to silt size particles predominantly $CaSO_3$, $CaSO_4$, metal oxides and silicates	Same as I	Same as I
Amount of Waste (kg/MT Product)	Acid Plant sludge - 122 kg Retort Scrubber Sludge - 10 kg ("blue powder")	Acid Plant Sludge - 122 kg Retort Scrubber Sludge - 0 if recycled	Same as II
Factors Affecting Hazardousness	Contains heavy metals, including As, Pb, Zn, Cd, Cr, Se, Hg	Same as I	Same as I
Treatment/ Disposal Technology	Unlined lagoons with dredged sludge stored or disposed on land	Unlined lagoons; immediate recycle of retort scrubber sludge; storage or disposal of dredged acid plant sludge on land	Lined Lagoons; immediate recycle of retort scrubber sludge; chemical fixation of acid plant sludge before land disposal
Estimate of # + % of Plants Using Technology	3 (75%)	1 (25%)	0

Table 36-(cont'd.)

SLUDGES (Continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Possible leaching or runoff of heavy metals	Same as I	Adequate
Problems and Comments	-	-	May be difficult to line lagoons in use
Non-Land Environmental Impact	Possible contamination of ground of surface water	Same as I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	Same as I	Groundwater and runoff water monitoring
Energy Requirements	Negligible	Negligible	Negligible

3.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Electrolytic Reduction Plants. Two lagoons are used for the settling of solids. One is relatively small, 2000 m^3; the other, 11,440 m^3, is sized to allow four days of residency. Characteristics of the lagoons are shown below:

	Lagoon	
	A	B
Bottom width	14 m	39 m
Top width	26 m	51 m
Bottom length	27 m	78 m
Top length	39 m	90 m
Total depth	3 m	3 m
Depth of excavation	1.75 m	1.1 m
Circumference	142 m	295 m
Dike Volume	975 m^3	3,808 m^3
Dike Surface	1,761 m^2	4,071 m^2
Total width	37 m	65 m
Total length	50 m	104 m
Required area	.2 ha	.7 ha

The lagoons are constructed on rural land.

The daily inflow into Lagoon A is 1,480 m^3 which contain 9.1 MT of solids. These solids are estimated to form 12 m^3 of sludge daily.

The daily inflow into Lagoon B is 2,860 m^3 which contain 17.5 MT of solids. These solids are estimated to form 23 m^3 of sludge.

Both lagoons are dredged periodically. The sludge is deposited next to the lagoons, allowed to dry and then recycled. Only the cost of lagoon dredging is included.

The lagoons are assumed dredged at 25 day intervals. The amount of sludge removed each time totals 875 m^3 or 12,250 m^3 for the year. A 3/4 yard drag line is employed. It has an hourly capacity of 27 m^3 (1). Consequently about 460 hours (25%) of equipment operating time are required annually. Costs of Level I treatment and disposal technology are given in Table 37.

Pyrometallurgical

The annual capacity of the plant is 107,000 MT operating 350 days/year. Effluents from primary gas cleaning operations and acid blowdown totaling 2,400 m^3/day go to a lagoon. The lagoon is designed for 4 day residency. Its design characteristics are:

TABLE 37

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY
PRIMARY ZINC PLANT - ELECTROLYTIC REDUCTION

Capital Cost

	Sludge (from acid plant anode washings, spent acid & bleeds)
Lagoon	
Site Preparation	
Survey	$ 125
Test Drilling	490
Sample testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	1,295
Compacting	1,805
Fine Grading	790
Soil Poisoning	175
Transverse Drain Fields	340
Land	350
Lagoon	
Site Preparation	
Survey	440
Test Drilling	490
Sample Testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	5,065
Compacting	7,045
Fine Grading	1,830
Soil Poisoning	365
Transverse Drain Fields	600
Land	1,225
Equipment	
Dragline (25%)	17,500
Total	$42,830

TABLE 37 (continued)

Annual Cost

	Sludge
Land	$ 160
Amortization	
Construction	2,760
Equipment	2,780
Operating Personnel	5,590
Repair and Maintenance	
Construction	580
Equipment	875
Energy	
Fuel	525
Electricity	55
Taxes	40
Insurance	430
Total	$13,795

Volume	9,600 m^3
Bottom width	36 m
Top width	48 m
Bottom length	71 m
Top length	83 m
Total depth	3 m
Depth of excavation	1.2 m
Circumference	273 m
Dike volume	3,241 m^3
Dike surface	3,713 m^2
Total width	61 m
Total length	96 m
Required area	.6 ha

The lagoon is located on rural land since primary zinc smelters are located in rural areas.

The daily lagoon inflow contains 37 MT of solids. These are estimated to result in the formation of 48.7 m^3 of sludge. It is assumed that the lagoon is dredged when the sludge accumulation equals 50 percent of the lagoon volume. Thus the lagoon is dredged every 99 days or 3.5 times/year. This requires 625 hr/yr of dragline time. The sludge is piled adjacent to the lagoon, dried, and then hauled to the retort residue dump.

A small portion of the zinc product vapor produced in the retorts passes through the zinc condensers and is recovered as "blue powder" from wet scrubber bleed slurry settled in two concrete pits. Each pit measures about 12 x 6 x 2.5 m. Wall thickeness is about .4 m requiring 36 x 2.5 x .4 m or 34.2 m^3 of concrete in place for wall construction.

About 3.1 MT of blue powder is dredged daily from the pits, stored on the ground for drying and oxidation and eventually recycled. One hour of backhoe time is assigned daily for this task. The average amount stored at any given time is about 535 MT.

Approximately 16,800 m^3 of dried sludge are generated annually. Transport requirements to load and haul the sludge to a dump are estimated to be 895 hours of truck and front loader time.

The dump area, providing space for the accumulation of 20 years of dried sludge extends over 3.4 hectares. The dump is located on rural land. About 200 hours of bulldozer time are assigned at the dump for spreading and compacting the sludge.

A samll amount of waste, about 0.5 MT/day, is generated by the associated cadmium plant. This waste is presently disposed with the wastes from the retort furnaces. Its disposal cost is negligible.

Cost of Level I treatment and disposal are given in Table 38.

TABLE 38

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY ZINC PLANT - PYROMETALLURGICAL REDUCTION

Capital Cost

Lagoon	Sludge
Site Preparation	
Survey	$ 375
Test Drilling	490
Sample Testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	4,310
Compacting	5,995
Fine Grading	1,670
Soil Poisoning	340
Transverse Drain Fields	560
Land	1,050
Sumps	19,360
Equipment	
Dump Truck (50%)	12,500
Frontloader (65%)	13,000
Dragline (35%)	24,500
Bulldozer (10%)	1,600
Dump	
Survey	2,025
Land	5,950
Total	$95,175

Annual Cost

Land	$ 700
Amortization	
Construction	4,245
Equipment	8,205
Operating Personnel	33,730
Repair and Maintenance	
Construction	965
Equipment	2,580
Energy	
Fuel	3,765
Electricity	375
Taxes	175
Insurance	950
Total	$55,690

3.4.2 Cost of Best Technology Currently Employed (Level II)

Electrolytic Reduction Plants. The technology and costs for Level II are the same as Level I.

Pyrometallurgical Plants. The blue powder dredged from the pit is recycled without prolonged ground storage. A 20 x 20 m concrete pad is constructed adjacent to the pit to prevent leaching during the temporary storage. The cadmium containing waste is immediately sent to a lead smelter. Associated capital and annual costs are shown below.

Capital Costs	
Concrete Pad	$ 6,140
Total	$ 6,140
Annual Cost	
Construction Amortization	$ 710
Construction Repair & Maint.	185
Insurance	60
Total	$ 955

Cost of Technology To Provide Adequate Health and Environmental Protection (Level III)

Electrolytic Reduction Plants. Level III consists of lining both lagoons. Additionally, the lagoon sludges are pumped into concrete pits prior to being recycled. This procedure eliminates the requirement of the dragline used for lagoon dredging in Level I and its attendant costs. The pits are sufficient to accommodate 25 days of accumulated sludge.

Net incremental costs are shown in Table 39. The pit dimensions are 3 x 6 x 10 m and 3 x 10 x 20 m, respectively. The slurry pump has a capacity of 236 l/min and is operated 740 hrs/yr; 1000 hrs/yr of labor are assigned. The assumed, required piping consists of 200 m of installed pipe and 100 m of flexible pipe, both 7.6 cm in diameters.

Pyrometallurgical Plants. The lagoon is lined and the lagoon sludge is chemically fixed. The costs are the same for rail and truck transport of the residue. Costs are given in Table 40.

Costs of Levels I, II and III treatment and disposal technology are summarized in Table 41 for electrolytic plants and in Table 42 for pyrometallurgical plants. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of zinc product. Costs for each level of technology and each type of waste are also expressed as a percent of metal selling price.

TABLE 39

COSTS OF LEVEL III TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY ZINC PLANT - ELECTROLYTIC REDUCTION

Capital Cost

Construction	Sludge
Lagoon A Liner	$ 6,020
Lagoon B Liner	22,515
Concrete Pit (Lagoon A)	11,440
Concrete Pit (Lagoon B	21,420
Equipment	
Slurry Pump	13,730
Pipe Rigid (Installed)	4,425
Pipe Flexible	440
(Dragline)	(24,500)
Total	$55,490

Annual Cost

Land	-
Amortization	
Construction	$ 7,130
Equipment 1/	(940)
Operating Personnel 1/	4,560
Repair and Maintenance	
Construction	1,840
Equipment 1/	(295)
Energy	
Fuel 1/	-
Electricity	(700)
Taxes	-
Insurance 1/	555
Total	$12,150

1/ Costs shown are not costs, i.e. costs of new equipment less dragline associated costs which are no longer incurred.

TABLE 40

COST OF LEVEL III TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY ZINC PLANT PYROMETALLURGICAL REDUCTION

CAPITAL COST

	Sludge
Construction	
Lagoon Liner	$ 19,800
Total	$ 19.800

ANNUAL COST

	Sludge
Land	-
Amortization	
Construction	$ 2,295
Equipment	-
Operating Personnel	-
Repair and Maintenance	
Construction	595
Equipment	-
Energy	
Fuel	-
Electricity	-
Taxes	-
Insurance	200
Chemical Fixation	221,760
Total	$224,850

TABLE 41 COST SUMMARY FOR ELECTROLYTIC ZINC SMELTING AND REFINING

Annual Production: Model Plant 100,000 MT Industry 187,001

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.15

Cum. Unit Waste Disposal Costs:

	Level								
Waste (Type)	I			II			III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$2.91	$0.97	$0.43	$2.91	$0.97	$0.43	$7.16	$2.39	$1.05
Annual Cost	0.94	0.31	0.14	0.94	0.31	0.14	1.87	0.62	0.27
Total Capital Cost	-		0.43	-		0.43	-		1.05
Total Annual Cost	-		0.14	-		0.14	-		0.27

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$0.08	$0.03	$0.08	$0.03	$0.20	$0.05
Total	0.08	0.03	0.08	0.03	0.20	0.05

1973 Metal Price: $454.52/MT

Treatment Cost As a Percent of Metal Selling Price

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.09%	0.03%	0.09%	0.03%	0.23%	0.06%
Total	0.09	0.03	0.09	0.03	0.23	0.06

TABLE 42 COST SUMMARY FOR PYROMETALLURGICAL ZINC SMELTING AND REFINING

Annual Production: Model Plant 107,000 MT Industry 305,107 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.19

Cum. Unit Waste Disposal Costs:

	Level								
Waste (Type)	I			II			III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$4.72	$1.57	$0.89	$5.03	$1.68	$0.95	$6.01	$2.00	$1.13
Annual Cost	2.76	0.92	0.52	2.81	0.94	0.53	13.96	4.65	2.63
Total Capital Cost	-		0.89	-		0.95	-		1.13
Total Annual Cost	-		0.52	-		0.53	-		2.63

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$0.27	$0.16	$0.29	$0.16	$0.34	$0.80
Total	0.27	0.16	0.29	0.16	0.34	0.80

1973 Metal Price: $454.52/MT

Treatment Costs As a Percent of Metal Selling Price

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.20%	0.11%	0.21%	0.12%	0.25%	0.58%
Total	0.20	0.11	0.21	0.12	0.25	0.58

Total estimated industry annualized costs for treatment and disposal of sludge from electrolytic zinc production in 1973 dollars are given in Table 41 while the estimated 1973 costs for sludge disposal from the pyrometallurgical process are given in Table 42. The annualized industry cost for Level I and Level II treatment and disposal of sludge from electrolytic zinc smelters is $30,000 or 0.03% of national sales. The annualized cost for Level III treatment and disposal technology (i.e. adequate for environmental protection is $50,000 or 0.06% of national sales.

The annualized industry cost for Levels I and II treatment and disposal technology for sludge from pyrometallurgical zinc production is $160,000 or 0.11% of national sales. The annualized industry cost for Level III treatment and disposal technology is $800,000 or 0.58% of national sales.

4.0 PRIMARY SMELTING AND REFINING OF ALUMINUM (SIC 3334)

4.1 INDUSTRY CHARACTERIZATION

There are 31 primary aluminum plants in the United States owned by thirteen different firms. Major producers are Aluminum Company of America (ALCOA), Anaconda Aluminum Company, Kaiser Aluminum & Chemical Corporation and Reynolds Aluminum. Total U.S. plant capacity is 4,418,000 MT/year. Distribution of plants by capacities is shown in Table 43. Output from primary aluminum reduction cells is molten aluminum which is usually cast into pure and alloy ingots and billets before shipping for product manufacture. The total U.S. value of primary aluminum production is given as $2,206,440,000 for 1973 (Ref. 2).

Aluminum metal is manufactured by the Hall-Heroult process, which involves the electrolytic reduction of alumina dissolved in a molten salt bath of cryolite (a complex of $\mathrm{NaF \cdot AlF_3}$) and various salt additives:

$$\underset{\text{Alumina}}{\mathrm{Al_2O_3}} \xrightarrow{\text{Electrolysis}} \underset{\text{Aluminum}}{\mathrm{4Al}} + \underset{\text{Oxygen}}{\mathrm{3O_2}}$$

The electrolysis is performed in a carbon crucible housed in a steel shell, known as a "pot." The electrolysis employs the carbon crucible as the cathode (negative pole) and a carbon mass as the anode (positive pole). The three types of anode configurations used are: prebaked (PB), horizontal-stud Soderberg (HSS), and vertical-stud Soderberg (VSS).

The major portion of aluminum produced in the United States is processed in cells using prebaked anodes. In this type of cell, the anode consists of blocks that are formed from a carbon paste and baked in an oven prior to their use in the cell. These blocks--typically 14 to 24 per cell--are attached to metal rods and serve as replaceable anodes. As the reduction proceeds, the carbon in these blocks is gradually consumed at a rate of about 1 inch per day by reaction with the oxygen by-product.

The second most commonly used anode configuration is of the horizontal-stud Soderberg type. In this type of anode, pitch and carbon "paste" are added periodically at the top of the superstructure and the mixture is baked in place as it descends closer to the molten bath. The entire anode assembly is moved downward as the carbon is consumed. The cell anode is contained by aluminum sheeting and preformed channels through which electrode connections, called studs, are inserted into the anode paste. As the baking anode is lowered, the lower row of studs and the bottom channel are removed, and the electrical connectors are moved to a higher row.

Table 43

GEOGRAPHIC DISTRIBUTION OF PRIMARY ALUMINUM PLANTS

STATE	NO. OF PLANTS	PLANT DISTRIBUTION BY CAPACITY (MT)			PLANT DISTRIBUTION BY PROCESS					CAPACITY PER PROCESS (MT)				
		30K-100K	100K-200K	200K-300K	B	SV	SH	SHB	SVB	B	SV	SH	SHB	SVB
ALABAMA	2	0	1	1	1	0	1	0	0	102000	0	200000	0	0
ARKANSAS	2	1	1	0	1	0	0	1	0	111000	0	0	57000	0
INDIANA	1	0	0	1	1	0	0	0	0	249000	0	0	0	0
KENTUCKY	2	0	2	0	2	0	0	0	0	272000	0	0	0	0
LOUISIANA	2	1	0	1	1	0	1	0	0	32000	0	236000	0	0
MARYLAND	1	1	0	0	1	0	0	0	0	79000	0	0	0	0
MISSOURI	1	1	0	0	1	0	0	0	0	64000	0	0	0	0
MONTANA	1	0	1	0	0	1	0	0	0	0	163000	0	0	0
NEW YORK	2	0	2	0	1	0	1	0	0	118000	0	116000	0	0
NORTH CAROLINA	1	0	1	0	1	0	0	0	0	104000	0	0	0	0
OHIO	1	0	0	1	1	0	0	0	0	218000	0	0	0	0
OREGON	2	1	1	0	1	1	0	0	0	118000	82000	0	0	0
TENNESSEE	2	0	1	1	1	0	0	0	1	127000	0	0	0	250000
TEXAS	3	0	2	1	1	1	1	0	0	254000	168000	101000	0	0
WASHINGTON	7	1	5	1	4	1	2	0	0	686000	108000	255000	0	0
WEST VIRGINIA	1	0	1	0	1	0	0	0	0	148000	0	0	0	0
EPA REGION														
I	0	0	0	0	0	0	0	0	0	0	0	0	0	0
II	2	0	2	0	1	0	1	0	0	118000	0	116000	0	0
III	2	1	1	0	2	0	0	0	0	227000	0	0	0	0
IV	7	0	5	2	5	0	1	0	1	605000	0	200000	0	250000
V	2	0	0	2	2	0	0	0	0	467000	0	0	0	0
VI	7	2	3	2	3	1	2	1	0	397000	168000	337000	57000	0
VII	1	1	0	0	1	0	0	0	0	64000	0	0	0	0
VIII	1	0	1	0	0	1	0	0	0	0	103000	0	0	0
IX	0	0	0	0	0	0	0	0	0	0	0	0	0	0
X	9	2	6	1	5	2	2	0	0	804000	190000	255000	0	0
NATION	31	6	18	7	19	4	6	1	1	2682000	521000	908000	57000	250000

B = PRE-BAKED SV = SODERBERG VERTICAL SH = SODERBERG HORIZONTAL
SHB = SODERBERG HORIZONTAL AND PRE-BAKED SVB = SODERBERG VERTICAL
AND PREBAKED

The vertical-stud Soderberg anode is similar to the horizontal-stud anode, with the exception that the studs are mounted vertically in the cell. The studs must be raised and replaced periodically. Representative raw material and energy requirements for aluminum reduction cells are presented in Table 44. A schematic representation of the reduction process is shown in Figure 6. Geographical distribution of U.S. primary aluminum plant capacity according to the types of processes employed (i.e., prebake, Soderberg) is given in Table 43.

4.2 WASTE CHARACTERIZATION

The hypothetical plant is assumed to have an annual capacity of 153,000 MT (169,000 short tons) of aluminum. This figure was obtained by averaging the 1973 production levels for those plants using wet-type primary air pollution control equipment and having a scrubbing liquor treatment system that is likely to require a settling basin or lagoon. Since aluminum reduction plants operate around the clock, 365 days a year, the daily capacity of the hypothetical plant is 420 MT (463 short tons).

4.2.1 Process Description

Essentially, all primary aluminum reduction plants use the Hall-Heroult electrolytic process (discussed previously) for producing free aluminum metal from alumina (Al_2O_3). The aluminum is produced in individual cells, which for the assumed typical plant total 700, each having a capacity of 600 Kg/day (1320 lb/day). The cells (or pots) are arranged in lines, called potlines, and are housed in potrooms. Fumes from the cells are collected by hoods located directly over the cells (primary control). In the hypothetical plant considered, the fumes collected by the hoods are scrubbed in a wet-type system. The assumed plant has no system for controlling the fumes that are not captured by the primary system. These uncollected fumes are vented to the atmosphere through ventilators located in the roofs of the potrooms.

Input materials for the potrooms consist of carbon anodes, carbon potliners (the cathodes), alumina (Al_2O_3), Cryolite (Na_3AlF_6), aluminum fluoride (AlF_3), and calcium fluoride (CaF_2). The anodes are prepared in the carbon plant where petroleum coke is crushed, screened, classified and mixed with a pitch binder. The mixture is pressed into the desired shape and baked. For Soderberg anodes, the pressing and baking operations are omitted, and the prepared paste is introduced directly into the anode. The hypothetical plant is of the prebaked anode configuration. The cathodes, or potliners, are also made in the carbon plant. Anthracite and pitch are used in their preparation.

When a sufficient quantity of aluminum metal has formed in a cell, it is drawn off into a transfer crucible and sent to the cast house where it is placed directly in a holding furnace or cast into pigs. Usually the molten metal is fluxed before casting to remove minor impurities.

Table 44

RAW MATERIAL AND ENERGY REQUIREMENTS FOR ALUMINUM PRODUCTION

Parameter	Representative value
Cell operating temperature	~1740°F (~950°C)
Current through pot line	60,000 to 125,000 amp
Voltage drop per cell	4.3 to 5.2
Current efficiency	85 to 90%
Energy required	6.0 to 8.5 kwh/lb aluminum (13.2 to 18.7 kwh/kg aluminum)
Weight alumina consumed	1.89 to 1.92 lb AL_2O_3/lb aluminum (1.89 to 1.92 kg AL_2O_3/kg aluminum)
Weight electrolyte fluoride consumed	0.03 to 0.10 lb fluoride/lb aluminum (0.03 to 0.10 kg fluoride/kg aluminum)
Weight carbon electrode consumed	0.45 to 0.55 lb electrode/lb aluminum (0.45 to 0.55 kg electrode/kg aluminum)

Source: Compilation of Air Pollutant Emission Factors,
2nd Ed., U.S.E.P.A., 1973 (AP-42)

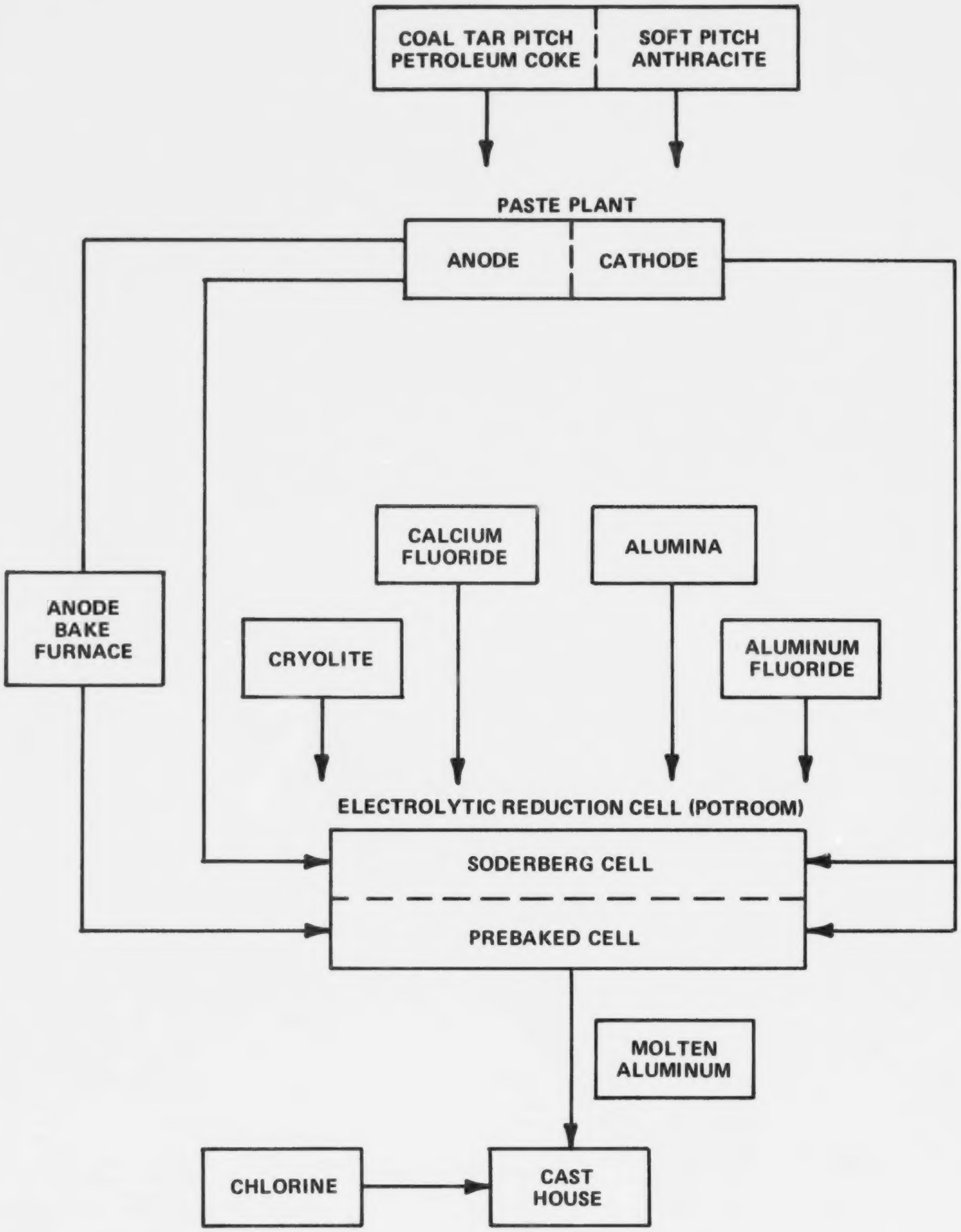

COAL TAR PITCH
PETROLEUM COKE
SOFT PITCH
ANTHRACITE
PASTE PLANT
ANODE
CATHODE
CALCIUM FLUORIDE
ALUMINA
ANODE BAKE FURNACE
CRYOLITE
ALUMINUM FLUORIDE
ELECTROLYTIC REDUCTION CELL (POTROOM)
SODERBERG CELL
PREBAKED CELL
MOLTEN ALUMINUM
CHLORINE
CAST HOUSE

Figure 6 PROCESS DIAGRAM FOR PRIMARY ALUMINUM SMELTING

4.2.2 Description of Waste Streams

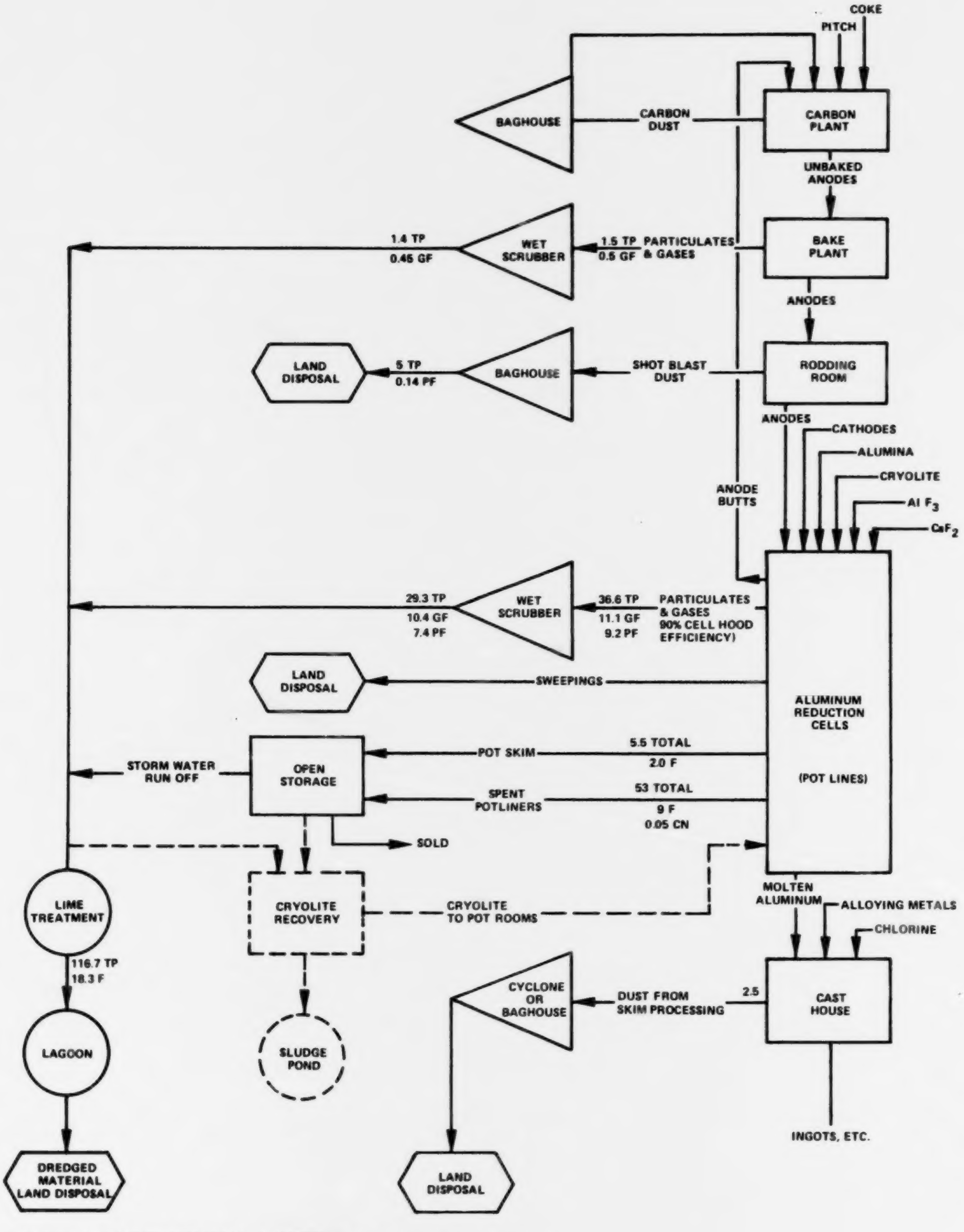
Compounds of fluorine are the major potentially hazardous materials found in the wastes generated in primary aluminum smelting. They are present in spent anode butts, in spent potlines, and in dusts deposited in the potrooms. At present, there are nine plants that employ dry systems for primary control of emissions. Of these nine plants, two employ wet systems for control of fugitive emissions which escape from furnaces into work areas. The remainder require no fugitive emissions controls. Seventeen plants have wet primary control systems, and at least four of these have wet fugitive emissions controls. The remainder have no fugitive emissions controls. There are four plants that control part of the potlines with dry primary systems and the remaining potlines with wet systems. None of the four plants have fugitive emissions controls. Figure 7 illustrates the aluminum smelting and refining processes and associated wastes.

Only those plants that employ prebaked anode require anode bake plants. Of 21 plants of the prebake type, four employ wet-type systems for control of bake plant emissions, three use dry-type systems, and nine have no bake plant emission control systems. The controls for the remaining five plants are not known.

Sludges

Emissions from the electrolytic cells amount to 36.6 Kg/MT aluminum and contain 11.1 Kg fluoride. Emissions from the anode bake furnace amount to 1.5 Kg/MT with 0.5 Kg fluoride. Wet scrubbers are assumed for emissions control in the electrolytic and anode bake furnaces. The scrubber waters are then lime treated resulting in sludges which are discharged or possibly recycled to the scrubbers. It is the treatment of this scrubber water used for primary air pollution control that accounts for the major portion of potentially hazardous sludge wastes generated at primary aluminum smelting plants. As shown in Figure 7 , 117 Kg of particulates per metric ton of molten aluminum containing 18 Kg of fluoride are discharged to lagoons after lime treatment. The lagoons are periodically dredged and dredged material disposed on land.

For the hypothetical plant wet scrubber,systems were assumed for emission control in the potrooms and the anode bake plant. In practice, the type of emission control varies widely within the industry. Some plants use dry-type control systems throughout, some use a mixture of wet and dry systems, and other use wet-type systems only. For example, sludge generated in the recovery of cryolite from the potroom scrubber water amounts to less than 20 Kg per metric ton of aluminum produced. In plants that recover cryolite from spent potliners, as well as from potroom scrubber water, sludge generation amounts to about 125 Kg per metric ton of aluminum. On the other hand, in plants that do not recover cryolite, sludge generated in the treatment of potroom scrubber water averages an estimated 95 Kg per metric ton of aluminum and can exceed 200 Kg per metric ton of aluminum for once-through lime treatment systems. In general, the use of dry systems reduces the amount of waste destined for disposal because dry collection, such as alumina bed adsorption allows direct recycle of captured materials. Baghouses are already


COKE
PITCH
CARBON PLANT
BAGHOUSE
CARBON DUST
UNBAKED ANODES
BAKE PLANT
WET SCRUBBER
1.5 TP
0.5 GF
PARTICULATES & GASES
1.4 TP
0.45 GF
ANODES
RODDING ROOM
BAGHOUSE
SHOT BLAST DUST
LAND DISPOSAL
5 TP
0.14 PF
ANODES
CATHODES
ALUMINA
CRYOLITE
Al F_3
CaF_2
ANODE BUTTS
ALUMINUM REDUCTION CELLS
(POT LINES)
WET SCRUBBER
36.6 TP
11.1 GF
9.2 PF
PARTICULATES & GASES 90% CELL HOOD EFFICIENCY)
29.3 TP
10.4 GF
7.4 PF
LAND DISPOSAL
SWEEPINGS
OPEN STORAGE
POT SKIM
5.5 TOTAL
2.0 F
SPENT POTLINERS
53 TOTAL
9 F
0.05 CN
STORM WATER RUN OFF
SOLD
CRYOLITE RECOVERY
CRYOLITE TO POT ROOMS
SLUDGE POND
LIME TREATMENT
116.7 TP
18.3 F
LAGOON
DREDGED MATERIAL LAND DISPOSAL
MOLTEN ALUMINUM
ALLOYING METALS
CHLORINE
CAST HOUSE
2.5
DUST FROM SKIM PROCESSING
CYCLONE OR BAGHOUSE
LAND DISPOSAL
INGOTS, ETC.

NOTE: ALL VALUES GIVEN AS Kg OF WASTE PER METRIC TON OF ALUMINUM PRODUCED.
TP = TOTAL PARTICULATES; F = FLUORINE; PF = PARTICULATE FLUORIDES;
GF = GASEOUS FLUORIDES; CN = CYANIDES

Figure 7 PRIMARY ALUMINUM PRODUCTION

commonly used in carbon plants, and the collected dust (mostly carbon) can be directly recycled in the anode preparation operations.

With respect to wet scrubber systems, it should be noted that there are several alternatives for treating the scrubber water. In the hypothetical plant, once-through lime treatment was assumed. Other alternatives include: recycling of the lime-treated scrubber water, and recycling of the scrubber water accompanied by recovery of the cryolite. The amount of sludge generated in scrubber water treatment systems that incorporate a cryolite recovery capability can be significantly less than that generated in once-through lime treatment systems.

Spent anode butts. During the electrolysis of the aluminum (Al_2O_3) to obtain molten aluminum, the carbon anodes are consumed to the point that they can no longer be used. At this point, the anode remnants or "butts" are broken up and sent back to the carbon plant for reclamation of carbon to prepare new anodes. Large chunks of anode butts are either immediately sent to the carbon plant for reprocessing or stored for short periods of time before reprocessing.

Spent potliners. Cathode potliners become partially saturated with cryolite (Na_3AlF_6). After continued use, they crack and the molten aluminum becomes contaminated with iron from the outer shelf of the cathode. At this point the cathodes must be replaced. Approximately 53 Kg of spent potliners containing 9 Kg of fluoride are produced per metric ton of molten aluminum. The spent potliners coated with cryolite are stored outside to await shipment to a purchaser who will process them to recover cryolite or to await later processing by the smelter itself if facilities are available. Storage may be for periods of many months to a year or more.

Potroom skimmings. Potroom skimmings are residues skimmed from the hot metal after tapping. These residues, consisting almost entirely of bath material, are generated at the rate of about 5.5 Kg per metric ton of aluminum produced and contain about 2.0 Kg of fluorides per metric ton of aluminum. Generally, the skimmings are returned directly to the pots. In some cases, the skimmings are handled in the same manner as spent potliners (i.e., stored and reprocessed).

Shotblasting dust. In the recovery of spent anodes, the steel rods in the center of the anodes are sand blasted to remove attached carbon and cryolite. Dust from this operation is generated at a rate of about 5 kilograms per metric ton of aluminum and contains 0.5 Kg of fluoride. This waste is normally landfilled.

Skim processing dust. Alloying furnaces in the cast houses produces a skim on the surface of the molten aluminum metal. This skim is removed and processed for metal recovery in small furnaces. Dust emissions collected from this operation amounts to approximately 2.5 Kg/MT of aluminum product.

Chemical analyses of waste samples obtained from primary aluminum plants are given in Appendix A . Spent potliners consist of large and small chunks of carbon coated with cryolite. In solubility tests as described in Appendix B , high concentrations of fluoride and cyanide were found to leach from spent potliners. Potliners are therefore considered potentially hazardous. Although solubility tests were not conducted on pot skimmings, soluble fluorides are expected to leach from them. Skimmings are therefore considered potentially hazardous.

Sludges originating from wet scrubbing of emissions from the anode bake plant or aluminum reduction cells were found to release soluble fluoride and/or cyanide in solubility tests described in Appendix B . They are therefore considered potentially hazardous.

Although shot blast dust was found to contain a high concentration of copper, very little was solubilized in solubility tests. It is not considered potentially hazardous. Cast house dust contains significant concentrations of the heavy metals Cu, Zn, Pb, Cr, and Ni. Solubility of these metals was not ascertained. Cast house is considered potentially hazardous at this time because of the high concentration of heavy metals and small particle sizes.

4.2.3 Waste Quantities

Table 45 summarizes the generation factors for each of the land-disposed or stored residuals in the primary aluminum smelting and refining category. This table also gives typical concentrations of potentially hazardous constituents. Table 46 gives the yearly quantity of wastes and potentially hazardous constituents for a typical plant producing 153,000 metric tons (169,000 short tons) of aluminum per year.

Using the waste generation factors and concentration factors contained in Table 45 and smelter capacities, estimates have been made for the quantities of land-disposed or stored residuals and hazardous constituents thereof. These estimates are given for 1974, 1977, and 1983 on a state-by-state, regional and national level in Table 47.

Projections for increased quantities of potliners and dust for 1977 were based on planned plant expansions as given in the March 1975 issue of Engineering and Mining Journal. Projections for 1983 were computed from the 1977 data assuming a four percent yearly growth indicated by representatives from the industry.

Our projections for 1977 and 1983 also take into account the trend toward dry emission control systems. Note that the projected total amounts of sludge for 1977 and 1983 are less than the national total for 1974. The 1977 projections were based on published information on plant expansions. Because similar information was not available for 1983, the 1983 levels for ducts and potliners were computed from the 1977 data assuming a four percent yearly growth. No increase in sludge amounts are shown for 1983 over 1977 because all expansions are assumed to use dry systems for potroom emissions.

Table 45
WASTE RESIDUAL FACTORS - PRIMARY ALUMINUM SMELTING

TYPE OF WASTE	GENERATION FACTOR	CONCENTRATION FACTORS (ppm)			
	Kg/MT OF ALUMINUM	FLUORINE	CYANIDE	COPPER	LEAD
SLUDGE - POTLINE SCRUBBERS	113[a]	160,000[b]	–	–	–
SLUDGE - CARBON BAKE PLANT SCRUBBERS	3.70	120,000	–	–	–
DUST - CAST HOUSE SKIM PROCESSING	2.5	–	–	6,200	4,600
DUST - RODDING ROOM SHOT BLASTING	5	28,000	–	15,000	–
SPENT POTLINERS	53	160,000	1,050	–	–
SKIM FROM ELECTROLYTIC CELLS	5.5	350,000	–	–	–

[a] VALUE FOR INDIVIDUAL PLANTS MAY RANGE FROM 30 TO 200 Kg/MT DEPENDING ON PLANT OPERATION AND LEVEL OF TREATMENT OF SCRUBBER WATER. HIGHEST SLUDGE VALUES CORRESPOND TO ONCE-THROUGH LIME TREATMENT AT 2X TO 3X STOICHIOMETRIC REQUIREMENT.

[b] FLUORINE CONCENTRATION VARIES WITH AMOUNT OF SLUDGE GENERATED. CONCENTRATION GENERALLY LOWER FOR LARGER SLUDGE VOLUMES.

Table 46

YEARLY GENERATION OF WASTE RESIDUALS - PRIMARY ALUMINUM SMELTING BY TYPICAL PLANT*

TYPE OF WASTE	TOTAL QUANTITY OF WASTE (MT)	QUANTITIES OF POTENTIALLY HAZARDOUS CONSTITUENTS (MT)			
		FLUORINE	CYANIDE	COPPER	LEAD
SLUDGE - POTLINE SCRUBBERS	17,290	2,770	–	–	–
SLUDGE - CARBON BAKE PLANT SCRUBBERS	566	67.9	–	–	–
DUST-CAST HOUSE SKIM PROCESSOR	383	–	–	2.4	1.8
DUST - RODDING ROOM SHOT BLASTING	765	21.4	–	11.5	–
SPENT POTLINERS	8,110	1,300	8.5	–	–
SKIM FROM ELECTROLYTIC CELLS	842	295	–	–	–

BASED ON PREBAKE ANODE PLANT WITH WET SCRUBBERS ON POTLINES AND CARBON BAKE PLANT.

*QUANTITY CALCULATED FROM GENERATION AND CONCENTRATION FACTORS GIVEN IN TABLE 45 BASED ON ANNUAL PRODUCTION OF 153,000 MT. DIVIDE BY 365 TO OBTAIN DAILY QUANTITIES. MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

Table 47-a

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ALUMINUM PRODUCTION 1974 (METRIC TONS, DRY WEIGHTS) *

STATE	SLUDGES						POTLINERS AND POTROOM SKIMMINGS[1]						DUSTS						
	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS		DISPOSAL METHOD[2]	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS		DISPOSAL METHOD	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS			DISPOSAL METHOD[3]
				FLUORIDE	CYANIDE					FLUORIDE	CYANIDE					FLUORIDE	COPPER	LEAD	
ALABAMA	33,718	33,718	2,830	2,830	–[4]	LAGOON	5,893	5,893	1,067	1,061	5.59	OPEN DUMP	1,261	1,261	30	14	13	3	OPEN DUMP
ARKANSAS	177	177	19	19	0	LAGOON	6,419	6,419	1,163	1,157	6.09	OPEN DUMP	1,044	1,044	33	18	13	2	OPEN DUMP
INDIANA	0	0	0	0	0	–	14,417	14,417	2,611	2,597	13.70	OPEN DUMP	1,864	1,864	62	35	24	3	OPEN DUMP
KENTUCKY	12,819	12,819	2,572	2,572	–	LAGOON	15,186	15,186	2,647	2,632	15.00	OPEN DUMP	2,041	2,041	67	38	26	3	OPEN DUMP
LOUISIANA	6,935	6,935	1,191	1,191	0.32	LAGOON	14,225	14,225	2,352	0	14.72	OPEN DUMP	1,193	1,193	34	24	7	3	OPEN DUMP
MARYLAND	1,539	1,539	77	77	0.12	LAGOON	0	0	0	0	0	–	592	592	20	11	8	1	OPEN DUMP
MISSOURI	0	0	0	0	0	–	3,736	3,736	677	673	3.54	OPEN DUMP	484	484	16	9	6	1	OPEN DUMP
MONTANA	10,369	10,369	3,919	3,919	0	LAGOON	6,664	6,664	1,560	1,551	8.98	OPEN DUMP	408	408	5	0	3	2	OPEN DUMP
NEW YORK	16,530	16,530	1,604	1,604	0.17	LAGOON	6,840	6,840	833	826	6.49	OPEN DUMP	2,087	2,087	29	17	11	1	OPEN DUMP
N. CAROLINA	0	0	0	0	0	–	6,051	6,051	1,096	1,090	5.72	OPEN DUMP	782	782	26	15	10	1	OPEN DUMP
OHIO	9,362	9,362	2,428	2,428	–	LAGOON	12,628	12,628	2,287	2,275	12.00	OPEN DUMP	1,633	1,633	56	31	21	3	OPEN DUMP
OREGON	4,000	4,000	1,200	1,200	–	LAGOON	4,899	4,899	694	694	<0.01	OPEN DUMP	885	885	29	17	11	1	OPEN DUMP
TENNESSEE	8,000	8,000	1,762	1,762	–	LAGOON	21,888	21,888	3,965	3,944	20.77	OPEN DUMP	2,646	2,646	82	46	32	4	OPEN DUMP
TEXAS	27,032	27,032	6,157	6,157	–	LAGOON	24,005	24,005	4,272	4,249	23.22	OPEN DUMP	2,577	2,577	70	36	28	6	OPEN DUMP
WASHINGTON	68,811	68,811	6,833	6,833	0.13	LAGOON	25,561	25,561	4,249	4,230	18.52	OPEN DUMP	8,711	8,711	371	294	68	9	OPEN DUMP
W. VIRGINIA	0	0	0	0	0	–	8,576	8,576	1,553	1,545	8.14	OPEN DUMP	1,109	1,109	37	21	14	2	OPEN DUMP
EPA REGION																			
II	16,530	16,530	1,604	1,604	0.17		6,840	6,840	833	826	6.49		2,087	2,087	29	17	11	1	
III	1,539	1,539	77	77	0.12		8,576	8,576	1,553	1,545	8.14		1,701	1,701	57	32	22	3	
IV	54,537	54,537	7,164	7,164	0		49,018	49,018	8,775	8,727	47.08		6,730	6,730	205	113	81	11	
V	9,362	9,362	2,428	2,428	0		27,045	27,045	4,896	4,872	25.70		3,497	3,497	117	66	45	6	
VI	34,144	34,144	7,367	7,367	0.12		44,649	44,649	7,787	7,743	44.03		4,814	4,814	137	78	48	11	
VII	0	0	0	0	0		3,736	3,736	677	673	3.54		484	484	16	9	6	1	
VIII	10,369	10,369	3,919	3,919	0		6,654	6,654	1,560	1,551	8.98		408	408	5	0	3	2	
X	72,811	72,811	8,033	8,033	0.13		30,460	30,460	4,943	4,924	18.53		9,596	9,596	400	311	79	10	
NATIONAL TOTAL	199,292	199,292	30,592	30,592	0.74		176,978	176,978	31,026	30,861	162.49		29,317	29,317	966	626	295	45	

1) APPROXIMATELY 90% POTLINERS, 10% SKIMMINGS

2) SLUDGES ARE OFTEN DREDGED PERIODICALLY AND DISPOSED ON LAND

3) AT SOME LOCATIONS DUSTS ARE COVERED TO PREVENT BLOWING

4) A DASH IN THIS COLUMN INDICATES NOT ENOUGH INFORMATION FOR ESTIMATE. AMOUNT EXPECTED IS VERY SMALL.

*MULTIPLY SLUDGE QUANTITIES BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

Table 47-b
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ALUMINUM PRODUCTION 1977 (METRIC TONS, DRY WEIGHTS)*

STATE	SLUDGES						POTLINERS AND POTROOM SKIMMINGS[1]						DUSTS						
	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS		DISPOSAL METHOD[2]	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS		DISPOSAL METHOD	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS			DISPOSAL METHOD
				FLUORIDE	CYANIDE					FLUORIDE	CYANIDE					FLUORINE	COPPER	LEAD	
ALABAMA	33,718	33,718	2,830	2,830	–[4]	LAGOON	18,942	18,942	3,430	3,412	18.0	OPEN DUMP	2,948	2,948	86	46	34	6	OPEN DUMP
ARKANSAS	177	177	19	19	0	LAGOON	6,419	6,419	1,163	1,157	6.1	OPEN DUMP	1,044	1,044	33	18	13	2	OPEN DUMP
INDIANA	0	0	0	0	0	–	14,417	14,417	2,611	2,597	13.7	OPEN DUMP	1,864	1,864	62	35	24	3	OPEN DUMP
KENTUCKY	12,819	12,819	2,572	2,572	–	LAGOON	15,186	15,186	2,647	2,632	15.0	OPEN DUMP	2,041	2,041	67	38	26	3	OPEN DUMP
LOUISIANA	6,935	6,935	1,191	1,191	0.3	LAGOON	14,225	14,225	2,352	2,337	14.7	OPEN DUMP	1,193	1,193	34	24	7	3	OPEN DUMP
MARYLAND	1,539	1,539	77	77	0.1	LAGOON	0	0	0	0	0	OPEN DUMP	1,178	1,178	39	22	15	2	OPEN DUMP
MISSOURI	0	0	0	0	0	–	7,366	7,366	1,334	1,327	7.0	OPEN DUMP	953	953	31	18	12	1	OPEN DUMP
MONTANA	10,369	10,369	3,919	3,919	0	LAGOON	6,654	6,654	1,560	1,551	9.0	OPEN DUMP	408	408	5	0	3	2	OPEN DUMP
NEW YORK	5,968	5,968	664	664	0.2	LAGOON	9,997	9,997	1,217	1,207	9.5	OPEN DUMP	2,515	2,515	42	24	16	2	OPEN DUMP
N. CAROLINA	0	0	0	0	0	–	9,471	9,471	1,715	1,706	9.0	OPEN DUMP	1,225	1,225	41	23	16	2	OPEN DUMP
OHIO	9,362	9,362	2,428	2,428	0	LAGOON	12,628	12,628	2,287	2,275	12.0	OPEN DUMP	1,633	1,633	55	31	21	3	OPEN DUMP
OREGON	4,000	4,000	1,200	1,200	–	LAGOON	16,915	16,915	4,566	4,548	18.3	OPEN DUMP	2,157	2,157	70	40	27	3	OPEN DUMP
TENNESSEE	8,000	8,000	1,762	1,762	–	LAGOON	21,888	21,888	3,965	3,944	20.8	OPEN DUMP	2,646	2,646	82	46	32	4	OPEN DUMP
TEXAS	27,032	27,032	6,157	6,157	–	LAGOON	24,005	24,005	4,272	4,249	23.2	OPEN DUMP	2,577	2,577	70	36	28	6	OPEN DUMP
WASHINGTON	68,811	68,811	6,833	6,833	0.1	LAGOON	25,561	25,561	4,249	4,230	18.5	OPEN DUMP	8,711	8,711	371	294	68	9	OPEN DUMP
W. VIRGINIA	0	0	0	0	0	–	8,576	8,576	1,553	1,545	8.1	OPEN DUMP	1,109	1,109	37	21	14	2	OPEN DUMP
EPA REGION																			
II	5,968	5,968	664	664	0.2		9,997	9,997	1,217	1,207	9.5		2,515	2,515	42	24	16	2	
III	1,539	1,539	77	77	0.1		8,576	8,576	1,563	1,545	8.1		2,287	2,287	76	43	29	4	
IV	54,537	54,537	7,164	7,164	–		65,487	65,487	11,757	11,694	62.8		8,860	8,860	276	153	108	15	
V	9,362	9,362	2,428	2,428	0		27,045	27,045	4,898	4,872	25.7		3,487	3,487	117	66	45	6	
VI	34,144	34,144	7,367	7,367	–		44,649	44,649	7,787	7,743	44.0		4,814	4,814	137	78	48	11	
VII	0	0	0	0	0		7,366	7,366	1,334	1,327	7.0		953	953	31	18	12	1	
VIII	10,369	10,369	3,919	3,919	0		6,654	6,654	1,560	1,551	9.0		408	408	5	0	3	2	
X	72,811	72,811	8,033	8,033	–		42,476	42,476	8,815	8,778	36.8		10,868	10,868	441	334	95	12	
NATIONAL TOTAL	188,730	188,730	29,652	29,652	0.74		212,250	212,250	38,921	38,717	203		34,202	34,202	1,125	716	356	53	

1) APPROXIMATELY 90% POTLINERS, 10% SKIMMINGS

2) SLUDGES MAY BE DREDGED PERIODICALLY AND OPEN DUMPED OR LEFT PERMANENTLY IN LAGOONS

3) AT SOME LOCATIONS DUSTS ARE COVERED TO PREVENT BLOWING

4) A DASH IN THIS COLUMN INDICATES NOT ENOUGH INFORMATION FOR ESTIMATE. AMOUNT EXPECTED IS VERY SMALL.

*MULTIPLY SLUDGE QUANTITIES BY 3.0 TO CONVERT TO
APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 47-c
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ALUMINUM PRODUCTION 1983 (METRIC TONS, DRY WEIGHTS)*

STATE	SLUDGES						POTLINERS AND POTROOM SKIMMINGS[1]						DUSTS						
	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS		DISPOSAL METHOD[2]	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS		DISPOSAL METHOD	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	HAZARDOUS CONSTITUENTS			DISPOSAL METHOD[3]
				FLUORIDE	CYANIDE					FLUORIDE	CYANIDE					FLUORIDE	COPPER	LEAD	
ALABAMA	33,718	33,718	2,830	2,830	–[4]	LAGOON	23,962	23,962	4,340	4,317	23	OPEN DUMP	3,994	3,994	121	66	47	8	OPEN DUMP
ARKANSAS	177	177	19	19	0	LAGOON	8,120	8,120	1,471	1,463	8	OPEN DUMP	1,378	1,378	44	24	17	3	OPEN DUMP
INDIANA	0	0	0	0	0	–	18,237	18,237	3,303	3,286	17	OPEN DUMP	2,358	2,358	78	44	30	4	OPEN DUMP
KENTUCKY	12,819	12,819	2,572	2,572	–	LAGOON	19,369	19,369	3,404	3,385	19	OPEN DUMP	2,582	2,582	85	48	33	4	OPEN DUMP
LOUISIANA	6,935	6,935	1,191	1,191	0.3	LAGOON	18,338	18,338	3,096	3,078	18	OPEN DUMP	1,725	1,725	52	34	14	4	OPEN DUMP
MARYLAND	1,539	1,539	77	77	0.1	LAGOON	0	0	0	0	0	–	1,490	1,490	49	28	19	2	OPEN DUMP
MISSOURI	0	0	0	0	0	–	9,318	9,318	1,688	1,679	9	OPEN DUMP	1,205	1,205	39	23	15	1	OPEN DUMP
MONTANA	10,369	10,369	3,919	3,919	0	LAGOON	9,164	9,164	2,014	2,003	11	OPEN DUMP	733	733	15	6	7	2	OPEN DUMP
NEW YORK	5,968	5,968	664	664	0.2	LAGOON	12,646	12,646	1,538	1,526	12	OPEN DUMP	3,088	3,088	61	35	23	3	OPEN DUMP
N. CAROLINA	0	0	0	0	0	–	11,981	11,981	2,170	2,159	11	OPEN DUMP	1,550	1,550	51	29	20	2	OPEN DUMP
OHIO	9,362	9,362	2,428	2,428	0	LAGOON	15,974	15,974	2,893	2,878	15	OPEN DUMP	2,066	2,066	69	39	26	4	OPEN DUMP
OREGON	4,000	4,000	1,200	1,200	–	LAGOON	21,263	21,263	3,659	3,643	16	OPEN DUMP	2,963	2,963	96	55	37	4	OPEN DUMP
TENNESSEE	8,000	8,000	1,762	1,762	–	LAGOON	27,689	27,689	5,015	4,989	26	OPEN DUMP	3,396	3,396	107	60	42	5	OPEN DUMP
TEXAS	27,032	27,032	6,157	6,157	–	LAGOON	30,490	30,490	5,446	5,417	29	OPEN DUMP	3,643	3,643	105	56	41	8	OPEN DUMP
WASHINGTON	68,811	68,811	6,833	6,833	0.1	LAGOON	32,281	32,281	5,464	5,440	24	OPEN DUMP	10,781	10,781	439	333	94	12	OPEN DUMP
W. VIRGINIA	0	0	0	0	0	–	10,849	10,849	1,965	1,955	10	OPEN DUMP	1,403	1,403	46	26	18	2	OPEN DUMP
EPA REGION																			
II	5,968	5,968	664	664	0.2		12,646	12,646	1,538	1,526	12		3,088	3,088	61	35	23	3	
III	1,539	1,539	77	77	0.1		10,849	10,849	1,965	1,955	10		2,893	2,893	95	54	37	4	
IV	54,537	54,537	7,164	7,164	–		83,001	83,001	14,929	14,850	79		11,522	11,522	364	203	142	19	
V	9,362	9,362	2,428	2,428	0		34,211	34,211	6,196	6,164	32		4,424	4,424	147	83	56	8	
VI	34,144	34,144	7,367	7,367	–		56,948	56,948	10,013	9,958	55		6,746	6,746	201	114	72	15	
VII	0	0	0	0	0		9,318	9,318	1,688	1,679	9		1,205	1,205	39	23	15	1	
VIII	10,369	10,369	3,919	3,919	0		9,164	9,164	2,014	2,003	11		733	733	15	6	7	2	
X	72,811	72,811	8,033	8,033	0.1		53,544	53,544	9,123	9,083	40		13,744	13,744	535	388	131	16	
NATIONAL TOTAL	188,730	188,730	29,652	29,652	0.7		269,681	269,681	47,466	47,218	248		44,355	44,355	1,457	906	483	68	

1) APPROXIMATELY 90% POTLINERS, 10% SKIMMINGS

2) SLUDGES MAY BE DREDGED PERIODICALLY AND OPEN DUMPED OR LEFT PERMANENTLY IN LAGOONS

3) AT SOME LOCATIONS DUSTS ARE COVERED TO PREVENT BLOWING

4) A DASH IN THIS COLUMN INDICATES NOT ENOUGH INFORMATION FOR ESTIMATE. AMOUNT EXPECTED IS VERY SMALL

*MULTIPLY SLUDGE QUANTITIES BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

4.3 TREATMENT AND DISPOSAL TECHNOLOGY

4.3.1 Current Waste Treatment and Disposal Practices

Potliners and Pot Skimmings. Currently spent potliners and pot skimmings are stored on the ground before cryolite recovery either on site or at an outside location.

Sludges. Sludges originating from lime treatment of pot room emissions scrubwater, carbon plant scrubwater, and potliner storage pile runoff are settled in unlined lagoons and may or not be periodically dredged with permanent dispostion in open dumps.

Dusts. Dust from shot blasting and skim processing is currently open dumped.

As discussed previously all of the above wastes are considered potentially hazardous. Levels of technology for their treatment and disposal and evaluation of environmental adequacy thereof are discussed in the next sections.

4.3.2 Present Treatment and Disposal Technology (Level I)

Potliners and Pot Skimmings. Spent potliners and pot skimmings are stored on the ground for months to years before cryolite recovery either on site or at an outside location. This practice is inadequate because of possible fluoride and cyanide leaching.

Sludges. For the hypothetical plant, scrubber water from the potrooms is combined with scrubber water from the carbon plant and runoff from the potliners storage pile. The combined stream is then treated with lime and pumped to an unlined lagoon. The lagoon deposits primarily contain carbon particles, alumina, fluorine and calcium (from the lime treatment). Most of the fluoride and calcium will occur as calcium fluoride. Periodically, the bottom deposits in the lagoon may be pumped out or clammed out in order to allow continued use of the lagoon. At some plants sludge is left permanently in lagoons and new lagoons built as lagoons are filled to capacity. These deposits amounting to some 117 kg/MT aluminum are disposed of on land, probably adjacent to the lagoon. When the material is first pumped it is thick and watery. Eventually it dries out into a firm black mass. Level I Technology for disposal of lagoon dredgings is disposal in open dumps or permanent disposal in lagoons.

Many of the primary aluminum plants are using the above methods or variations thereof for treatment and disposition of process and pollution control residuals. Fluorides and cyanides could present environmental threats if they leach through permeable soils to groundwater or are carried in overland water runoff to surface streams.

Dusts. Dust from shot blasting(rodding room) and skim processing (cast house) is open dumped. As discussed previously shot blast dust did not leach in solubility tests and open dumping is considered adequate. Cast house dust is suspected of leaching and open dumping therefore is not considered adequate.

4.3.3 Best Technology Currently Employed (Level II)

Pot Liners and Pot Skimmings. A segment of the primary aluminum industry immediately processes spent potliners and skimmings for cryolite recovery without storing on the ground for any significant period of time. This practice will preclude any leaching or runoff of fluorides or cyanides and is therefore considered as Level II technology. It is noted, however, that there is a trend in the industry toward dry-type cell emission control systems. In plants having such systems, sodium fluoride emitted by the cells is intercepted by the emission control system, along with gases and other particulates, and returned to the cells, As a result of this recovery process, the loss of sodium fluoride is greatly reduced and recovery of aluminum fluoride from spent potliners is of greater interest than the recovery of cryolite. Thus cryolite could become in excess supply and be of little or no value.

Sludges. At the present time no primary aluminum plants are using lined or sealed sludge disposal areas. Therefore, Level II technology is the same as Level I for containment of sludges from the carbon plant and cryolite recovery. This technology may not be adequate for ground and surface water protection if leaching or runoff of fluorides is significant. Dry alumina absorption beds are used at some plants for the collection of reduction cell emissions as previously described. The collected emissions are recycled directly to the potlines thus precluding sludge generation from pot lines.

Dusts. Open dumping of shot blast dust from anode plants and skim process dust from cast houses is Level II technology since more desirable practices are not used.

4.3.4 Technology To Provide Adequate Health and Environmental Protection (Level III)

Potliners and Pot Skimmings. The practice of processing spent potliners and pot skimmings for cryolite recovery without significant storage periods on the ground provides adequate environmental protection and will therefore qualify as Level III technology.

Sludges. If significant leaching of fluorides from lagoon sludges through permeable soils to groundwater occurs, then the use of lined lagoons is needed as Level III. In addition the land area upon which dredged lagoon sediments are disposed should be treated with bentonite or other sealant. Alternatively the use of dry alumina absorption beds for collection and recycle of emissions from pot liners precludes lagoon or other land disposal practices.

Dusts. Sealing of ground areas where dust from skim processing (cast house) is disposed is needed if trace metals (Cu, Zn, Pb, Cr, Ni) leach through permeable soils to groundwater. Open dumping of shot blast dust is considered adequate at present in view of insignificant leaching of trace metals in solubility tests.

Tables 48 through 50 summarize features of Levels I, II and III treatment and disposal technologies for primary aluminum smelters including assessment of the hazardous nature of residuals and the methods and adequacy of treatment and disposal methods.

4.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. The hypothetical plant is assumed to have an annual capacity of 153,000 MT of aluminum per year and operates 365 days/year.

4.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Scrubber liquor for potrooms and the carbon plant together with run-off water from the potliner storage pile and other water are lime treated and put into a lagoon. The daily inflow is 105,980 m^3. Lagoon characteristics to accommodate this flow for adequate retention time are:

Volume	425,000 m^3
Bottom width	262 m
Top width	274 m
Bottom length	523 m
Top length	535 m
Total depth	3 m
Depth of excavation	.25 m
Circumference	1,630 m
Dike volume	38,088 m^3
Dike surface	25,591
Total width	291 m
Total length	552 ha
Required area	16 ha

Eight test holes are drilled and 16 soil samples analyzed. The lagoon is situated on rural land.

Approximately 42 MT of solids a day or 17,900 MT/yr flow into the lagoon daily. These form approximately 113 m^3 of sludge. The sludge is removed by pumping. A 2HP slurry pump with a capacity of 236 l/min is employed. The sludge is pumped through a 7.6 cm pipe to a sludge dump. The flow rate is 0.85 m/sec. The sludge dump is located some distance from the lagoon. Pipe costs include 200 m of installed pipe and 300 m of flexible pipe. Pumping operations are conducted on the average of 9.5 hr/day.

Table 48

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary Aluminum SIC 3334

SCRUBBER SLUDGES (carbon bake plant, potliners)

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Aggregated colloidal and silt size particles, carbon, AlF_3, Na_3AlF_6, CaF_2	No sludge to be disposed of	a) Same as I for Lime Prec. sludge b) Same as II for cryolite recovery
Amount of Waste (kg/MT Product)	117 kg (Lime Precipitated Sludge)	0	a) 117 kg (Lime precipitated sludge) b) 0 (cryolite recovery)
Factors Affecting Hazardousness	Leaching of fluoride	No sludge	a) For Lime Prec. Sludge, chance of leaching reduced b) For cryolite recovery, no sludge
Treatment/ Disposal Technology	Unlined lagoons	Cryolite recovery	a) Lined lagoons with sealing of sludge disposal areas, or b) cryolite recovery
Estimate of # + % of Plants Using Technology	14 (45%)	9 (29%)	a) Lined lagoons or land sealing - none b) cryolite recovery with unlined lagoon for residues - 9 (29%)

Table 48 - (cont'd.)

SCRUBBER SLUDGES

Factor	Level I	Level II	Level III
Adequacy of Technology	May leach fluorides	Adequate	Adequate
Problems and Comments	-	Quality of recovered cryolite may not meet specifications of all users; may not be economical	-
Non-Land Environmental Impact	Possible contamination of ground or surface water	None	None
Compatibility With Existing Facilities	Compatible	Large plant or group of plants needed for economy of cryolite recovery	Same as II
Monitoring & Surveillance Methods	None	None needed (No sludge)	a) Wells for detecting leaks from lined lagoons b) None required
Energy Requirements	Small	Moderate	a) Zero for lined lagoons b) Moderate for cryolite recovery

Table 49

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary Aluminum SIC 3334

SPENT POTLINERS AND SKIMMINGS

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Skimmings-crusted chunks of aluminum metal with carbon and cryolite impurities Potliners-large and small chunks of carbon coated with cryolite (Na_3AlF_6),	Sludge containing carbon and fluoride	Same as II
Amount of Waste (kg/MT Product)	Pot Liners-53 kg (stored) Skimmings-5.5 kg	70 kg of sludge	16 kg of sludge
Factors Affecting Hazardousness	May leach fluorides and traces of cyanides	No hazard if immediately recycled	No hazard if immediately recycled
Treatment/ Disposal Technology	Ground storage	Immediate recycle for cryolite and carbon recovery	Immediate recycle for cryolite and carbon recovery (improved system)
Estimate of # + % of Plants Using Technology	28 *	4	1

Table 49 - (cont'd.)

SPENT POTLINERS AND SKIMMINGS (continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	May leach fluoride and cyanide	Adequate	Adequate
Problems and Comments	* Includes 2 plants that recycle potliners for cryolite recovery and have storage piles	Quality of recovered cryolite may not meet specifications for all users; may not be economical	Same as Level II
Non-Land Environmental Impact	Possible contamination of ground or surface water	None	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Monitor runoff from storage piles for fluoride, cyanide
Energy Requirements	None	Not known	Not known

Table 50

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary Aluminum SIC 3334

Shot Blast Dust, Cast House Dust

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Shot blast dust-carbon, iron plus fluorides Cast House Dust-metal and metal oxides (predominantly Al)	Same as I	Same as I
Amount of Waste (kg/MT Product)	Shot blast dust - 5 kg Cast house dust - 2.5 kg	Same as I	Same as I
Factors Affecting Hazardousness	Shot blast dust - possible fluoride leaching Cast House Dust - possible leaching of heavy metals	Same as I	Same as I
Treatment/ Disposal Technology	Open Dump	Open Dump	Sealed soil area for dumping
Estimate of # + % of Plants Using Technology	31 (100%)	31 (100%	0 (0%)

Table 50-(cont'd.)

Shot Blast Dust, Cast House Dust, (cont.d)

Factor	Level I	Level II	Level III
Adequacy of Technology	May leach fluorides, heavy metals	Same as I	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	May contaminate ground or surface water	Same as Level I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	Same as Level I	Test wells in dump area; runoff water monitoring
Energy Requirements	Negligible	Negligible	Negligible

About 41,390 m^3 of sludge are generated annually which amounts to 827,800 m^3 over 20 years. The sludge, when dried, reduces in volume by 40 per cent, thus requiring storage for about 500,000 m^3 of waste. The area requirement is 5 ha assuming sludge build-up to a height of 10 m. The area is initially excavated to a depth of .5 m to prevent excessive run-off. Twelve hours of labor time/day is assigned to the pumping operation.

Solid waste consisting of discarded potliners is generated at a rate of 22 MT/day or 8030 MT/yr based on 52.5 Kg/MT of aluminum produced. The potliners have an estimated density of 2404 Kg/M^3. The potliners are eventually recycled, however, the average amount stored represents 3 years of production which amounts to about 10,000 m^3 (24,000 MT) of waste. This requires an area of about .25 ha for storage assuming the potliners are piled to a height of 5 m.

One hour/day of frontleader and truck time is allocated for transporting the potliners to the dump.

Other sources of solid waste are dusts from shot blasting and processing of cast house skim. Approximately 540 MT of shot blasting dust and 330 MT of skim processing dust are produced annually. The densities of the dusts are 1285 Kg/m^3 and 1045 Kg/m^3, respectively. This represents 870 m^3 of waste which are disposed on land each year. Loading and transporting this material requires 48 hours of frontloader and 48 hours of truck time each year. An area measuring about 0.2 ha is needed to store 20 years of waste given that the waste is piled to a 10 m height.

Capital and annualized costs of Level I treatment and disposal are summarized in Table 51. The annual cost per metric ton of aluminum produced is estimated as $0.62.

4.4.2 Cost of Best Technology Currently Employed (Level II)

The use of lined lagoons is not presently practiced in the industry and thus is not costed as Level II technology. Level II sludge and dust disposal is the same as Level I.

Potliners are recycled within a few weeks. This eliminates the need for the potliner dump and reduces the proportionate use of the truck from 20 to 2 1/2 percent. A 20 x 20 m concrete pad is provided for temporary storage of 2 - 4 days accumulation of used potliners. The resultant incremental costs are:

Capital Cost

Concrete pad	$ 6,140
Potliner dump	(595)
Dump truck	(4,375)
Total	$ 1,170

TABLE 51

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY ALUMINUM PLANT

Capital Cost

	Sludge	Potliners	Dust
Lagoon			
Site Preparation			
Survey	$10,000		
Test Drilling	1,970		
Sample Testing	1,000		
Report Preparation	2,400		
Construction			
Excavation & Forming	50,655		
Compacting	70,465		
Fine Grading	11,515		
Soil Poisoning	2,020		
Transverse Drain Fields	24,095		
Land	28,000		
Dump			
Survey	3,125	$ 155	$ 125
Land	8,750	440	350
Excavation & Forming	33,250		
Equipment			
Slurry Pump	13,730		
Pipe, Rigid (installed)	4,425		
Pipe, Flexible	1,320		
Dump Truck (20%)		4,300	700
Frontloader (20%)		3,440	560
Total	$266,720	$8,335	$1,735

Annual Cost

	Sludge	Potliners	Dust
Land	$3,675	45	35
Amortization			
Construction	24,415	20	15
Equipment	3,095	1,230	200
Operating Personnel	42,130	7,885	1,035
Repair and Maintenance			
Construction	4,765	-	-
Equipment	975	385	65
Energy			
Fuel	-	1,000	130
Electricity	80	100	15
Taxes	920	10	10
Insurance	2,665	85	15
Total	$82,720	$10,760	$1,520

Annual Cost

Land	$ (45)
Construction Amortization	645
Equipment Amortization	(695)
Operating personnel	
Construction maintenance and repair	165
Equipment maintenance and repair	(220)
Energy	
Fuel	(585)
Taxes	(10)
Insurance	10
Total	(735)

4.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection (Level III).

In addition to immediate recycling of potliners, the lagoon is lined, the ground is sealed at the sludge and dust disposal areas, and collection ditches, pumps and piping installed. Costs incurred are shown in Table 52.

Costs of Levels I, II and III treatment and disposal technologies are summarized in Table 53 for sludge, potliners and dust residuals from primary aluminum smelting and refining. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of aluminum product. Costs for each type of waste and each level of technology are also expressed as a percent of metal selling price.

Total industry costs for Levels, I, II and III treatment and disposal technologies have been estimated in 1973 dollars and are presented in Table 53. The annualized industry cost for Level I and II treatment and disposal technologies is estimated as 2,550,000 or 0.09% of national sales.

The annualized industry cost for Levels III which is adequate for environmental protection is $5,720,000 or 0.21% of estimated national sales. The large increase in cost from Level II to II is a result of using lined lagoons for scrubber sludge.

TABLE 52

COST OF LEVEL III TREATMENT AND DISPOSAL TECHNOLOGY
PRIMARY ALUMINUM PLANT

Capital Cost

	Sludge	Potliners	Dust
Construction			
Lagoon Liner	$609,795	-	
Land Sealing	100,000	-	$4,000
Collection Ditches 1/	9,360	-	355
Equipment			
Pump and Piping 2/	14,625	-	585
Total	$733,780	-	$4,940

Annual Cost

	Sludge	Potliners	Dust
Land	-	-	-
Amortization	$	-	
Construction	$ 83,420	-	$ 505
Equipment	2,325	-	95
Operating Personnel	-	-	-
Repair and Maintenance			
Construction	21,575	-	130
Equipment	730	-	30
Energy			
Fuel	-	-	-
Electricity	250	-	10
Taxes	-	-	-
Insurance	7,340	-	50
Total	$115,640		$ 820

1/ Ditches are installed around perimeter of sludge dump and 1.5 x 1 side of dust disposal area.

2/ Common pump and piping is used for run-off from sludge and dust dump.

TABLE 53 COST SUMMARY FOR PRIMARY ALUMINUM SMELTING AND REFINING

Annual Production: Model Plant 153,000 MT Industry 4,117,273 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.35
Potliners	0.05
Dust	0.006

Cum. Unit Waste Disposal Costs:

Waste (Type)	Level I $/MT of Waste Dry	Level I $/MT of Waste Wet	Level I $/MT of Prod.	Level II $/MT of Waste Dry	Level II $/MT of Waste Wet	Level II $/MT of Prod.	Level III $/MT of Waste Dry	Level III $/MT of Waste Wet	Level III $/MT of Prod.
Sludge Capital Cost	$4.96	1.65	$1.74	$4.96	1.65	$1.74	$18.60	6.20	$6.54
Sludge Annual Cost	1.54	0.51	0.54	1.54	0.51	0.54	3.69	1.23	1.30
Potliners Capital Cost	1.04		0.05	1.19		0.06	1.19		0.06
Potliners Annual Cost	1.34		0.07	1.25		0.07	1.25		0.07
Dust Capital Cost	1.99		0.01	1.99		0.01	7.67		0.04
Dust Annual Cost	1.75		0.01	1.75		0.01	2.69		0.02
Total Capital Cost	-		1,80	-		1.81	-		6.64
Total Annual Cost	-		0.62	-		0.62	-		1.39

Cum. Industry Waste Disposal Costs ($ Million)

Waste (Type)	Level I Cap.	Level I Ann.	Level II Cap.	Level II Ann.	Level III Cap.	Level III Ann.
Sludge	$7.16	$2.22	$7.16	$2.22	$26.93	$5.35
Potliners	0.21	0.29	0.25	0.29	0.25	0.29
Dust	0.04	0.04	0.04	0.04	0.16	0.08
Total	7.41	2.55	7.45	2.55	27.34	5.72

TABLE 53 (cont.) COST SUMMARY FOR PRIMARY ALUMINUM SMELTING AND REFINING

Metal Price: $638.00/MT

Treatment Cost As a Percent of Metal Selling Price

Waste (Type)	Level					
	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.27%	0.08%	0.27%	0.08%	1.02%	0.20%
Potliners	0.008	0.01	0.009	0.01	0.009	0.01
Dust	0.002	0.002	0.002	0.002	0.006	0.003
Total	0.28	0.09	0.28	0.09	1.03	0.21

5.0 PRIMARY SMELTING AND REFINING OF ANTIMONY (SIC 3339)

5.1 INDUSTRY CHARACTERIZATION

The primary antimony industry is relatively small. Production of antimony from domestic smelters in 1972 was 12,106 MT of which only 3,480 MT was antimony metal (Ref. 1). The remainder consisted of oxides and sulfides of antimony byproduct, antimonial lead, and other antimony residues. The selling price of antimony metal in 1973 was given as 59 cents/lb. Thus national sales are estimated at $4,528,000 (Ref. 2).

Plants producing antimony metal are the NL Industries, Inc. smelter at Laredo, Texas, the Sunshine Mining Co. electrolytic plant at Big Creek, Idaho, and the U.S. Antimony Corporation plant at Thompson Falls, Montana. Some antimony is recovered as a lead smelter byproduct (~7% of production in 1972) at three primary lead smelters. Over 90% of byproduct antimony produced at primary lead smelters was consumed at the smelter in manufacturing antimonial lead.

Approximately 80% of domestic antimony is produced at the Laredo, Texas smelter. Most of the remainder is produced electrolytically in Idaho. The U.S. Antimony Corporation produced 142 tons in 1972 (Ref. 1). Table 54 gives the geographical distribution of primary antimony plants by state, region, and nationally, according to process type.

The methods selected for the production of antimony metal or oxide are largely determined by the grade of the ore or concentrates. The lowest grade ores or concentrates (5 to 25 percent antimony) are generally treated by volatilization processes; those containing 25 to 40 percent antimony are smelted in a blast furnace. High-grade ores containing 45 to 60 percent antimony may be treated by a process called liquation, in which the antimony is reduced, melted, and drained from the gangue material. Special concentrates, such as silver-antimony concentrate, are treated by leaching and electrowinning. Condensed oxides from volatilization processes, liquated antimony sulfide, and some intermediate-grade ores are smelted in reverberatory furnaces.

5.2 WASTE CHARACTERIZATION

5.2.1 Process Descriptions

Antimony metal is produced by pyrometallurgical and electrolytic processes as described below.

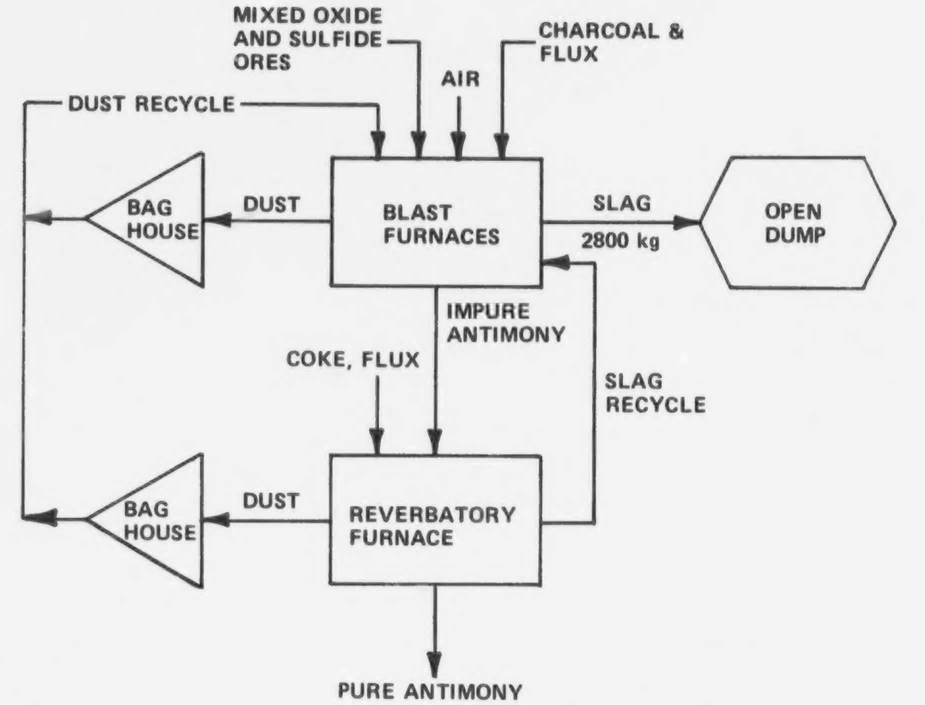
Pyrometallurgical

The operation at Laredo, Texas (Fig. 8) utilizies blast furnace smelting. The charge to the blast furnace consists of mixed oxide and sulfide ores from mines in Mexico or the United States. and byproducts of other smelting operations: mattes, slags, flue dusts, or residues from lead and zinc

TABLE 54

GEOGRAPHICAL DISTRIBUTION OF PRIMARY ANTIMONY METAL SMELTING AND REFINING

State	No. of Plants	Process Type	Estimated Annual Production (MT)
Idaho	1	electrolytic	900
Montana	1	pyrometallurgical	200
Texas	1	pyrometallurgical	2700
EPA REGION			
VI	1	pyrometallurgical	2700
VII	1	pyrometallurgical	200
X	1	electrolytic	900
NATIONAL	3		3800 MT


MIXED OXIDE AND SULFIDE ORES
AIR
CHARCOAL & FLUX
DUST RECYCLE
BAG HOUSE
DUST
BLAST FURNACES
SLAG
2800 kg
OPEN DUMP
IMPURE ANTIMONY
COKE, FLUX
SLAG RECYCLE
BAG HOUSE
DUST
REVERBATORY FURNACE
PURE ANTIMONY

NUMERICAL VALUE IS IN KILOGRAM/MT
OF ANTIMONY PRODUCED

Figure 8 PYROMETALLURGICAL ANTIMONY SMELTING AND REFINING

refineries. The blast-furnace operation is somewhat similar to that of lead blast-furnace practice. A vertical or shaft furnace with a rectangular cross section is used, with low-pressure air being supplied. The ore and residues, charcoal, and fluxing additions are charged at the top of the furnace and the products of molten antimony metals and slag are tapped from the hearth at the bottom of the furnace. The blast-furnace process is capable of producing extremely pure metal with minimal antimony loss.

Electrolytic

Electrolytic primary antimony metal is produced by a leaching-electrolysis process. The process is depicted in Figure 9 . In this process, a complex copper-antimony sulfide concentrate is leached with sodium sulfide solution which dissolves the antimony as sodium thioantimonate (Na_3SbS_4). The leach solution is clarified by settling and filtration, and electrolyzed in diaphragm cells to yield antimony of 93 to 99 percent purity. During the electrolysis, some sulfide sulfur is oxidized to thiosulfate or sulfate. With recirculation of the electrolyte, the concentration of these oxidized forms of sulfur would build up to the point where they would interfere with the solubilization of antimony in the leaching step. By treating spent electrolyte with barium sulfide, these sulfates are kept at acceptable levels; and the sulfide concentration is maintained. The barium sulfate and thiosulfate precipitate is filtered and heated with coal in a rotary kiln to reconstitute barium sulfide for reuse.

5.2.2 Description of Waste Streams

Pyrometallurgical

The only waste produced in the pyrometallurgical process for antimony metal production is a hard siliceous blast furnace slag produced at a rate of 2,800 Kg/MT of antimony metal. All other byproducts are recycled to the blast furnace. Analysis of a blast furnace slag from an antimony smelter is given in Appendix A . Antimony and zinc are present in high concentrations (i.e., Sb -18,000 ppm; Zn - 500 ppm). Leaching of trace metals from the large chunks of hard siliceous slag would not be anticipated. However, in solubility tests as described in Appendix B arsenic, antimony, copper, and zinc were found to leach in significant concentrations. Blast furnace slag is therefore considered potentially hazardous at this time.

Electrolytic

The only discard waste from electrolytic antimony production is spent anolyte solution which is discharged to a tailings pond which receives wastes from the mining and milling operations also. Approximately $13m^3$ of anolyte solution containing 210 Kg/MT of antimony metal is produced. The tailings pond will accumulate 190 MT of dry solids per year from the electrolytic operation.

Analyses of solids disposed of in tailings ponds is given in

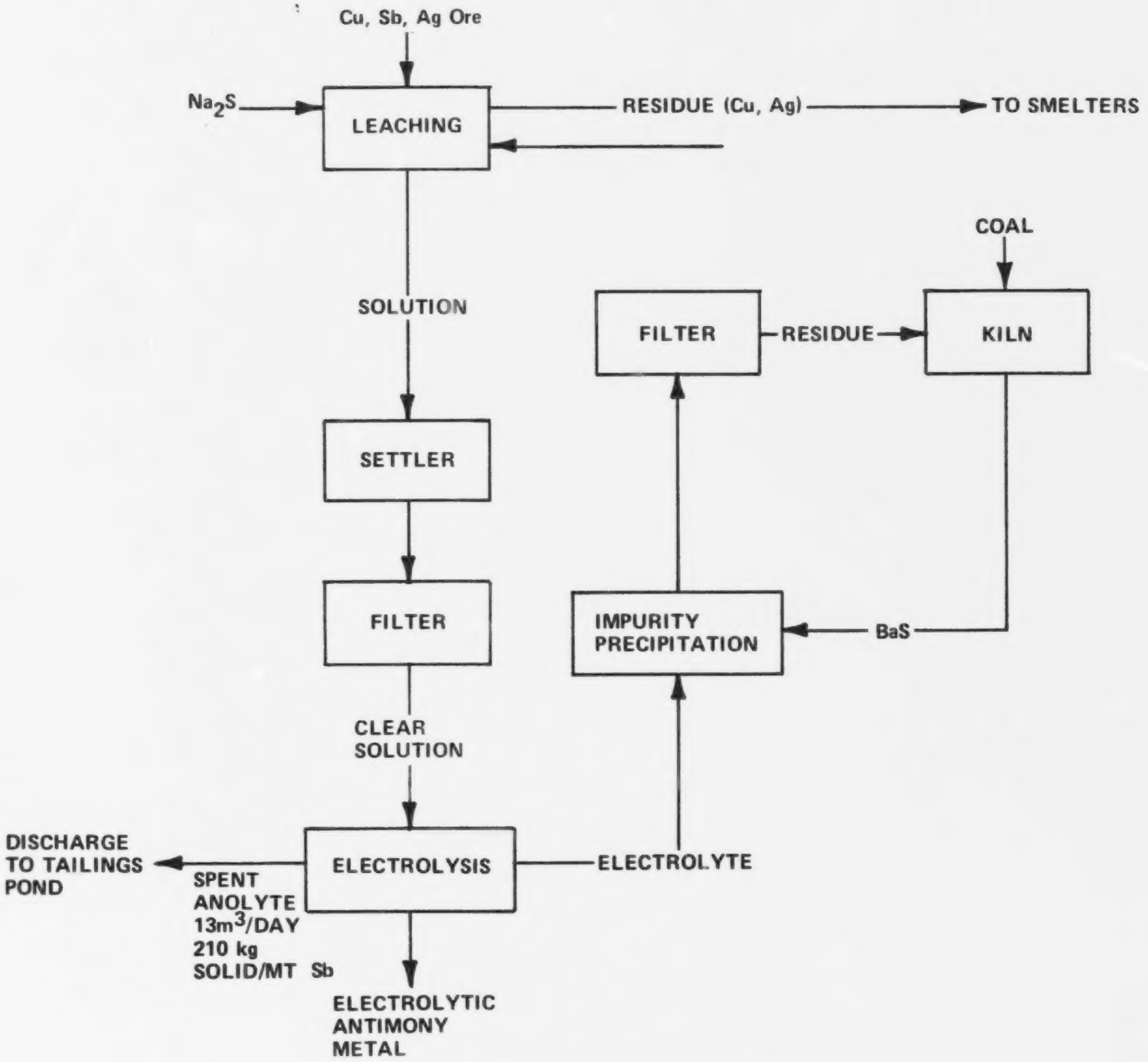

Cu, Sb, Ag Ore
Na_2S
LEACHING
RESIDUE (Cu, Ag)
TO SMELTERS
SOLUTION
SETTLER
FILTER
CLEAR SOLUTION
ELECTROLYSIS
DISCHARGE TO TAILINGS POND
SPENT ANOLYTE $13m^3$/DAY 210 kg SOLID/MT Sb
ELECTROLYTIC ANTIMONY METAL
ELECTROLYTE
IMPURITY PRECIPITATION
FILTER
RESIDUE
KILN
COAL
BaS

Figure 9 ELECTROLYTIC ANTIMONY REFINING

Appendix A. Solids contained traces of Pb, Cu, Zn, Ni, Mn, Cr, As and Cd. Antimony was the highest concentration (27,000 ppm). In solubility tests described in Appendix B, antimony, copper, cadmium and lead were found to leach to a minor extent. For this reason along with the very fine particulate nature of the sludge exposed to weathering processes, electrolytic sludge from primary antimony plants is considered potentially hazardous.

5.2.3 WASTE QUANTITIES

Table 55 summarizes the generation factors for each of the land disposed residuals in the primary antimony melting and refining category (i.e., blast furnace slag, electrolytic sludge). This table also gives typical concentration of potentially hazardous constituents as derived from the analysis of blast furnace slag and spent anolyte solids given in Appendix A. Table 56 gives the yearly quantity of wastes and potentially hazardous constituents for a typical pyrometallurgical plant producing 2900 MT/year of antimony metal. Table 56 gives yearly wastes for a typical plant producing 900 MT/year of antimony metal by the elctrolytic process.

Using the waste generation factors and concentrations contained in Tables 55 and plant capacities, estimates have been made for the quantities of land disposed residuals and hazardous constituents thereof. These estimates are given for 1974, 1977 and 1983 on a state-by-state-regional and national level in Table 57.

Primary antimony metal production has fluctuated from a low of 2594 MT in 1973 to a high of 3462 MT over the period of 1969 to 1973 (Ref. 2). Thus, no expansion in the industry can be predicted over the period 1974 to 1983 with waste generation based on a total original production of 2900 MT for each of the predictive years (1974, 1977, 1983). Texas is the largest producer of primary antimony and generates the largest quantity of potentially hazardous waste.

5.3 TREATMENT AND DISPOSAL TECHNOLOGY

5.3.1 Current Treatment and Disposal Technology

At the current time discard slag from smelting furnaces of pyrometallurgical plants is open dumped and spent anolyte solution from the electrolytic plant is discharged to the tailings pond. Since both of these wastes are considered potentially hazardous levels of technology for their treatment and disposal are discussed in more detail in the following sections.

5.3.2 Present Treatment and Disposal Technology (Level I)

Blast furnace/reverberatory slag. At the present time discard slags from reverberatory and/or blast furnaces from both pyrometallurgical plants are open dumped on site. This practice is inadequate if heavy metals leach through permeable soils to groundwater.

TABLE 55 WASTE RESIDUAL FACTORS - PRIMARY ANTIMONY SMELTING AND REFINING

Type of Waste	Generation Factor (Production) kg/MT	Concentration Factor (ppm) As	Pb	Cu	Zn	Ni	Sb	Cr	Cd
blast furnace/reverbatory slag (pyrometallurgical process)	2800	-	66	50	500	-	18,000	-	-
spent anolyte (electrolytic process)	210	16	5	50	2	5	27,000	32	1.0

TABLE 56 YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL PLANTS, PRIMARY ANTIMONY SMELTING AND REFINING

Type of Waste	Total Waste Quantity	Quantity of Potentially Hazardous Constituents (MT) As	Pb	Cu	Zn	Ni	Sb	Cr	Cd
Pyrometallurgical Plant:									
blast furnace/ reverbatory slag	7560	-	0.5	0.4	3.8	-	137	-	-
Electrolytic Plant:									
spent anolyte sludge	189	>.01	>0.01	>0.1	>.01	-	5.2	>0.1	>0.1

* Quantities calculated from generation and concentration factors given in Table 55 based on annual production of 2700 MT antimony metal by pyrometallurgical process plant and 900 MT by electrolytic plant. Divide by 365 to obtain daily quantities, multiply by 1.1 to convert to short tons.

Table 57-a

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ANTIMONY SMELTING AND REFINING TOTAL BLAST FURNACE SLAG - 1974, 1977, 1983 (METRIC TON)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS			
					Cu	Pb	Sb	Zn
MONTANA	560	560	10.4	OPEN DUMP	>0.1	>0.1	10.1	0.3
TEXAS	7560	7560	141.7	OPEN DUMP	0.4	0.5	137	3.8
EPA REGION VII	7560	7560	141.7		0.4	0.5	137	3.8
EPA REGION X	560	560	10.4		>0.1	>0.1	10.1	0.3
NATIONAL	8120	8120	152.1		0.4	0.5	147.1	4.1

*MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

Table 57-b

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY ANTIMONY SMELTING AND REFINING TOTAL ANOLYTE SLUDGE - 1974, 1977, 1983 (METRIC TON, DRY WEIGHT)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS
					Sb
IDAHO	189	189	5.2	TAILINGS POND	5.2
EPA REGION VI	189	189	5.2		5.2
NATIONAL	189	189	5.2		5.2

*MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Spent Anolyte Solution. At the present time spent anolyte solution from the one electrolytic antimony plant is discharged to the on-site unlined mill tailings pond. This practice is inadequate if heavy metals leach through permeable soils to groundwater.

5.3.3 Best Technology Currently Employed (Level II)

Blast Furnace/Reverberatory Slag. The practice and adequacy of Level II slag disposal is the same as Level I, since open dumping is the only practice employed.

Spent Anolyte Solution. The practice and adequacy of Level II spent anolyte solution disposal is the same as Level I since discharge to unlined tailings ponds is the only practice employed.

5.3.4 Technology Necessary To Provide Adequate Health and Environmental Protection (Level III)

Blast Furnace/Reverberatory Slag. In the event that leaching of heavy metals through permeable soils to groundwater occurs (i.e., Sb, Zn) sealing of ground at slag disposal areas with collection and treatment of runoff will be required to provide adequate health and environmental protection.

Spent Anolyte Solution. Sediments and water discharged to the tailings pond from the electrolytic antimony refining contain appreciable concentrations of antimony, sulfide, thallium, tellurium, and other potentially hazardous heavy metals. If significant leaching of contained hazardous metals from the fine sedimants occurs through permeable soils to water tables a separate lined lagoon should be constructed for permanent containment of sludges as Level III technology. Only treated water would be discharged to the main tailings pond. Tables 58 and 59 summarize features of Levels I, II and III treatment and disposal technology for pyrometallurgical and electrolytic refining of antimony including assessment of the hazardous nature of residuals and the methods and adequacy of treatment and disposal methods.

5.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. Pyrometallurgical plants and electrolytic plants will be considered separately because of differences in production technology and types and quantities of land disposed wastes.

5.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Pyrometallurgical Plant. The typical plant produces 2,700 MT of metal annually operating 365 days/year. In the process, 7,500 MT of slag are generated. The slag is deposited on an on-site dump.

One hour/day of front loader and backhoe time are assigned for

Table 58

Treatment and Disposal Technology Levels
PRIMARY (Pyrometallurgical)
Smelting and Refining Category ANTIMONY SIC 3339

BLAST FURNACE SLAG

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Large chunks of high density siliceous slag	Same as Level I	Same as Level I
Amount of Waste (kg/MT Product)	350 kg	350 kg	350 kg
Factors Affecting Hazardousness	Contains heavy metals including Sb, Pb, As Cu, Zn	Same as I	Same as I
Treatment/ Disposal Technology	Open dump	Open dump	Sealing of soil at disposal areas; collection and treatment of runoff
Estimate of # + % of Plants Using Technology	2 (100%)	2 (100%)	0 (0%)

Table 58 (cont'd.)
BLAST FURNACE SLAG - Continued

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if leaching of heavy metals occurs	Same as I	Adequate
Problems and Comments	Leaching of As, Cu, Sb, Zn in solubility tests (see Appendix)	same as I	None
Non-Land Environmental Impact	Possible contamination of ground and surface water if leached	Same as I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Monitoring wells and runoff monitoring
Energy Requirements	Negligible	Negligible	Negligible

Table 59

Treatment and Disposal Technology Levels

Smelting and Refining Category PRIMARY (Electrolytic) ANTIMONY SIC 3339

SPENT ANOLYTE SLUDGE

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Aggregated sludge of silt size particles, silicates, oxides	Same as I	Same as I
Amount of Waste (kg/MT Product)	210 kg	210 kg	210 kg
Factors Affecting Hazardousness	Contains heavy metals including Sb, As, Cu, Cr	Same as I	Same as I
Treatment/ Disposal Technology	Tailings pond	Tailings pond	Separate lined lagoon for sludge impoundment
Estimate of # + % of Plants Using Technology	1 (100%)	1 (100%	0 (0%)

Table 59 -(cont'd.)

SPENT ANOLYTE SLUDGE (Continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if heavy heavy metals are leached	Same as I	Adequate
Problems and Comments	Leaching of cadmium and antimony in solubility tests (see Appendix B.)	-	-
Non-Land Environmental Impact	Possible groundwater or surface water contamination if heavy metal leaching	Same as I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Monitoring wells
Energy Requirements	Negligible	Negligible	Negligible

hauling the slag to the dump. A dump area of about .9 ha is required to store slag produced over a 20 year period, assuming it is piled to a 10 m height. The dump is located on rural land and 45 hours/year of bulldozer time are assigned at the dump for spreading and compacting. The costs of Level I treatment and disposal technology are given in Table 60.

Electrolytic Plant. Annual production is 900 MT based on 365 days of operation.

Waste discharge consists of 13.2 m^3 of waste water daily containing 193 MT of solids per year which is discharged into a tailings pond. The tailings pond is associated with a mine-mill complex. The refinery waste represents less than 1 percent of the total discharge into the tailings pond. Consequently, no cost is assigned to the refinery discharge.

5.4.2 Cost of Best Technology Currently Employed (Level II)

Pyrometallurgical Plant. The technology and costs of Level II treatment and disposal of blast furnace slag, the only land disposed waste, is the same as Level I.

Electrolytic Plant. The technology and costs of Level II treatment and disposal of spent anolyte, the only land disposed waste, is the same as Level I.

5.4.3 Cost of Technology to Provide Adequate Health & Environmental Protection

Pyrometallurgical Plant. The land is sealed at the slag dump; collection ditches and pump and piping are installed. Additional costs incurred are given in Table 61 . Additional costs for this level amount to approximately one half of present costs (i.e., Level III costs are 1 1/2 times present costs). Since the amount of slag is large relative to the amount of product, costs are less per ton of waste than product produced.

Electrolytic Plant. The waste water from the electrolytic plant (spent anolyte) is directed to a lined settling pond before the overflow enters the tailings pond. About 528 Kg/day of dry solids are discharged into lagoon where they form .32 m^3 of sludge. The yearly sludge accumulation is 116.8 m^3.

The lagoon is designed to retain 20 years accumulation of sludge, (i.e. 2336 m^3), containing 3860 MT of dry solids or ∿11,000 MT of wet sludge a lagoon is generally effective until it is one half filled with sludge. Thus, the selected lagoon size to hold the volume of sludge generated in 20 years is 4,800 m^3. Its design characteristics are:

TABLE 60

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY ANTIMONY PLANT - PYROMETALLURGICAL REFINING

Capital Cost

	Reverbatory slag
Equipment	
Dump Truck (20%)	$ 5,000
Front Loader (20%)	4,000
Bulldozer (3%)	480
Dump	
Survey	560
Land	1,580
Total	$11,620

Annual Cost

Land	$ 160
Amortization	
Construction	65
Equipment	1,510
Operating Personnel	8,430
Repair and Maintenance	
Construction	-
Equipment	475
Energy	
Fuel	1,040
Electricity	105
Taxes	40
Insurance	115
Total	$11,940

TABLE 61 INCREMENTAL COSTS FOR LAND SEALING AND RUNOFF COLLECTION FOR SLAG DISPOSAL, PRIMARY ANTIMONY PRODUCTION

Capital Costs	
Land Sealing	$18,000
Collection Ditches	1,435
Pump & Piping	10,010
Total	$29,445
Annual Cost	
Construction Amortization	$ 2,255
Equipment Amortization	1,595
Construction Maintenance & Repair	585
Equipment Maintenance & Repair	500
Energy	
Electricity	50
Insurance	295
Total	$ 5,280

Bottom width	24 m	Circumference	201 m
Top width	36 m	Dike Volume	1,900 m^3
Bottom length	47 m	Dike Surface	2,615 m^2
Top length	59 m	Total Width	48 m
Total depth	3 m	Total Length	71 m
Depth of Excavation	1.45 m	Required Area	.35 ha

Costs are shown in Table 62. Since a new separate lined lagoon would be required for anolyte,all costs would be incremental.

Costs of Levels I, II and III treatment and disposal technologies for pyrometallurgical and electrolytic plants are summarized in Tables 63 and 64 Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of antimony product. Costs for each level of technology are also expressed as a percent of metal selling price.

Total industry costs for Levels I, II, and III treatment and disposal technologies have been estimated in 1973 dollars and are presented in Table 63 for pyrometallurgical antimony production and Table 64 for electrolytic antimony production. The annualized industry cost for Level I and II treatment and disposal of pyrometallurgical antimony waste slag is $10,000 or 0.29% of national sales value. The annualized cost of Level III technology (i.e., adequate for environmental protection) is 10,200 or 0.42% of national sales value.

Since sludge from electrolytic antimony production comprises approximately only one percent of a larger waste stream from a mine-mill complex, costs under Levels I and II treatment and disposal technology are negligible. The settling of the sludge in a separate lined lagoon before discharge to the main tailings pond results in an industry capital investment of ~ $10,000 and 2,000 annualized costs which represents 0.25% of national sales.

TABLE 62

COST OF LEVEL III TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY ANTIMONY PLANT - ELECTROLYTIC REFINING

Capital Cost

	Spent Anolyte Sludge
Lagoon	
Site Preparation	
Survey	$ 220
Test Drilling	490
Sample Testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	2,525
Compacting	3,515
Fine Grading	1,175
Soil Poisoning	250
Transverse Drain Fields	440
Lagoon Liner	11,190
Land	615
Total	$21,870

Annual Cost

Land	$ 60
Amortization	
Construction	2,470
Equipment	-
Operating Personnel	-
Repair and Maintenance	
Construction	575
Equipment	-
Energy	
Fuel	-
Electricity	-
Taxes	15
Insurance	220
Total	$ 3,340

TABLE 63 COST SUMMARY FOR PYROMETALLURGICAL ANTIMONY SMELTING AND REFINING

Annual Production: Model Plant 2,700 MT Industry 2,002 MT

Waste (Type)	Amount (MT/MT of Production)
Slag	2.8

Cumulative Unit Waste Disposal Costs:

	Level					
	I		II		III	
Waste (Type)	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.
Slag						
Capital Cost	$ 1.55	$4.30	$1.55	$4.30	$5.48	$15.21
Annual Cost	1.59	4.42	1.59	4.42	2.30	6.38
Total Capital Cost	-	4.30	-	4.30	-	15.21
Total Annual Cost	-	4.42	-	4.42	-	6.38

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
	I		II		III	
Waste (Type)	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	$0.01	$0.01	$0.01	$0.01	$0.03	$0.012
Total	0.01	0.01	0.01	0.01	0.03	0.012

1973 Metal Price: $1,507/MT

Treatment Cost As Percent of Metal Selling Price

	Level					
	I		II		III	
Waste (Type)	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	0.29%	0.29%	0.29%	0.29%	1.01%	0.42%
Total	0.29	0.29	0.29	0.29	1.01	0.42

TABLE 64 COST SUMMARY FOR PRIMARY ELECTROLYTIC ANTIMONY REFINING

Annual Production: Model Plant 900 MT Industry 501 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.21

Cumulative Unit Waste Disposal Costs:

Waste (Type)	Level I $/MT of Waste		Level II $/MT of Waste		Level III $/MT of Waste		
		$/MT of Prod.		$/MT of Prod.	Dry	Wet	$/MT of Prod.
Sludge							
Capital Cost	1/	1/	1/	1/	$113.32	$37.77	$24.30
Annual Cost	1/	1/	1/	1/	17.31	5.77	3.71
Total Capital Cost	-	1/	-	1/	-		24.30
Total Annual Cost	-	1/	-	1/	-		3.71

Cum. Industry Waste Disposal Costs ($ Million)

Waste (Type)	Level I Cap.	I Ann.	II Cap.	II Ann.	III Cap.	III Ann.
Sludge	1/	1/	1/	1/	$0.01	$.002
Total	1/	1/	1/	1/	$0.01	$.002

1973 Metal Price: $1507/MT

Treatment Cost As Percent of Metal Selling Price

Waste (Type)	Level I Cap.	I Ann.	II Cap.	II Ann.	III Cap.	III Ann.
Sludge	1/	1/	1/	1/	1.61%	0.25%
Total	1/	1/	1/	1/	1.61	0.25

1/ = negligible

6.0 PRIMARY SMELTING AND REFINING OF MERCURY (SIC 3339)

6.1 INDUSTRY CHARACTERIZATION

The primary mercury smelting and refining industry has been centered in California, Nevada, and Oregon. Mercury has also been recovered from ore in Arizona, Alaska, Idaho, Texas, and Washington and is recovered as a byproduct from gold ore in Nevada and zinc ore in New York. Due to low prices and slackened demand the mercury industry has been in a decline during recent years. Table 65 illustrates the decline in the industry (Ref. 3). In 1969 there were 109 producing mines. In 1973 there were only 6 producing mines. It has been predicted (Ref. 3), given such variables as market prices and effects of emission standards promulgated by EPA in April 1973, that production of primary mercury could range from a high of 20,000 flasks to a low of 3,000 flasks by 1985 but is expected to remain steady at about 3,000 flasks through 1985. Table 66 shows the geographical distribution of primary mercury production capacity as of late 1973. It must be kept in mind that operations open and close on short notice and resultant production can fluctuate greatly. U.S. sales of primary mercury was given as $1,601,000 in 1972 (Ref. 1).

TABLE 65. SALIENT DOMESTIC MERCURY STATISTICS

	1969	1970	1971	1972	1973
Producing Mines	109	79	56	21	6
Production-flasks*	29,640	27,296	17,883	7,286	2,200

* 1 flask = 34.47 kg (76 lb)

Mercury is extracted from ore and concentrate by heating in retorts or furnaces to liberate the metal as a vapor, followed by cooling of the vapor and collection of the condensed mercury. Recovery of mercury is high, averaging about 98 percent for retort installations and 95 percent for furnace plants.

6.2 WASTE CHARACTERIZATION

This section contains descriptions of production technology at primary mercury smelters and the resultant byproducts or wastes which are either recycled directly or disposed of on land. Estimates are given for the quantities of wastes and potentially hazardous constituents thereof which are disposed of on land.

6.2.1 Process Description

As discussed previously mercury is normally recovered directly from ores either in retorts or furnaces. Retorts are inexpensive installations for small operations and require only simple firing and condensing equipment. They are best adapted to operations treating 500 pounds to 5 tons per day of high-grade sorted ore.

Table 66

GEOGRAPHICAL DISTRIBUTION OF PRIMARY MERCURY METAL SMELTING AND REFINING CAPACITY

State	No. of Plants	Plant Distribution By Capacity (MT)		Plant Distribution By Process	Capacity Per Process (MT)
		<4	4-20	Pyrometallurgical	Pyrometallurgical
California	4	0	4	4	76
Nevada	1	1	0	1	4
EPA Region X	5	1	4	5	80
National	5	1	4	5	80

For larger operations, either rotary or multiple-hearth furnaces with mechanical feeding and discharging devices are preferred. Furnace capacities range from 10 to 100 MT per day with larger sizes to meet requirements.

Mercury-laden gases pass from the furnaces through dust collectors into the condensers, where the vapor is cooled and the mercury collected. Final traces of mercury in the gases from the condensers are removed in washers, and the stripped gases are discharged through a stack into the atmosphere. The mercury from the condenser always contains some dirt and soot, which are separated by hoeing the mixture with lime.

Mercury can also be leached from its ores and concentrates with a solution of sodium sulfide and sodium hydroxide and can be recovered as the metal by precipitation with aluminum or by electrolysis. Leaching of mercury ores has not been practiced extensively in the past. However use of these methods may increase in the future.

6.2.2 Description of Waste Streams

Calcine Residue. Figure 10 shows the various operations in pyrometallurgical recovery of mercury using retorts or furnaces along with the evolution and disposition of wastes. The largest source of waste is the furnace or retort residue amounting to some 207 MT per metric ton of metallic mercury produced. In solubility tests described in Appendix B leaching of toxic trace metals from furnace residue was insignificant (less than 0.5 ppm). Furnace or retort residues (i.e. calcine residues) are not considered potentially hazardous, therefore, at this time.

Condenser Cooling and Washing Water. There is a small volume of water (43 m^3/MT metal) discharged from condenser cooling and washdown. This amounts to about 5 m /day. Samples or analyses of condenser wastewater were not available but traces of free mercury are expected. This wastewater is therefore considered potentially hazardous.

As noted in Figure 10 dirty mercury may be filtered or cleaned to obtain a purer grade. Triple distillation may even be done for purification. In some cases nitric acid may be used for cleaning. In such cases the small laboratory scale volume of waste acid will contain soluble mercurous nitrate and should not be disposed on on land or in lagoons. All traces of mercury should be removed from the acid by distillation.

The typical plant representing the most prevalent plant size within the industry produces 500 flasks of mercury per year (76 lb/flask) or 17.3 MT per year. The typical plant operates about 150 days per year. Daily production of mercury is therefore about 0.115 MT of mercury. Daily discharge of condenser water is approximately 4.9 m^3 to a lagoon or the slag pile. There is a calcine residue of approximately 24 MT per day. Yearly discharge of water to

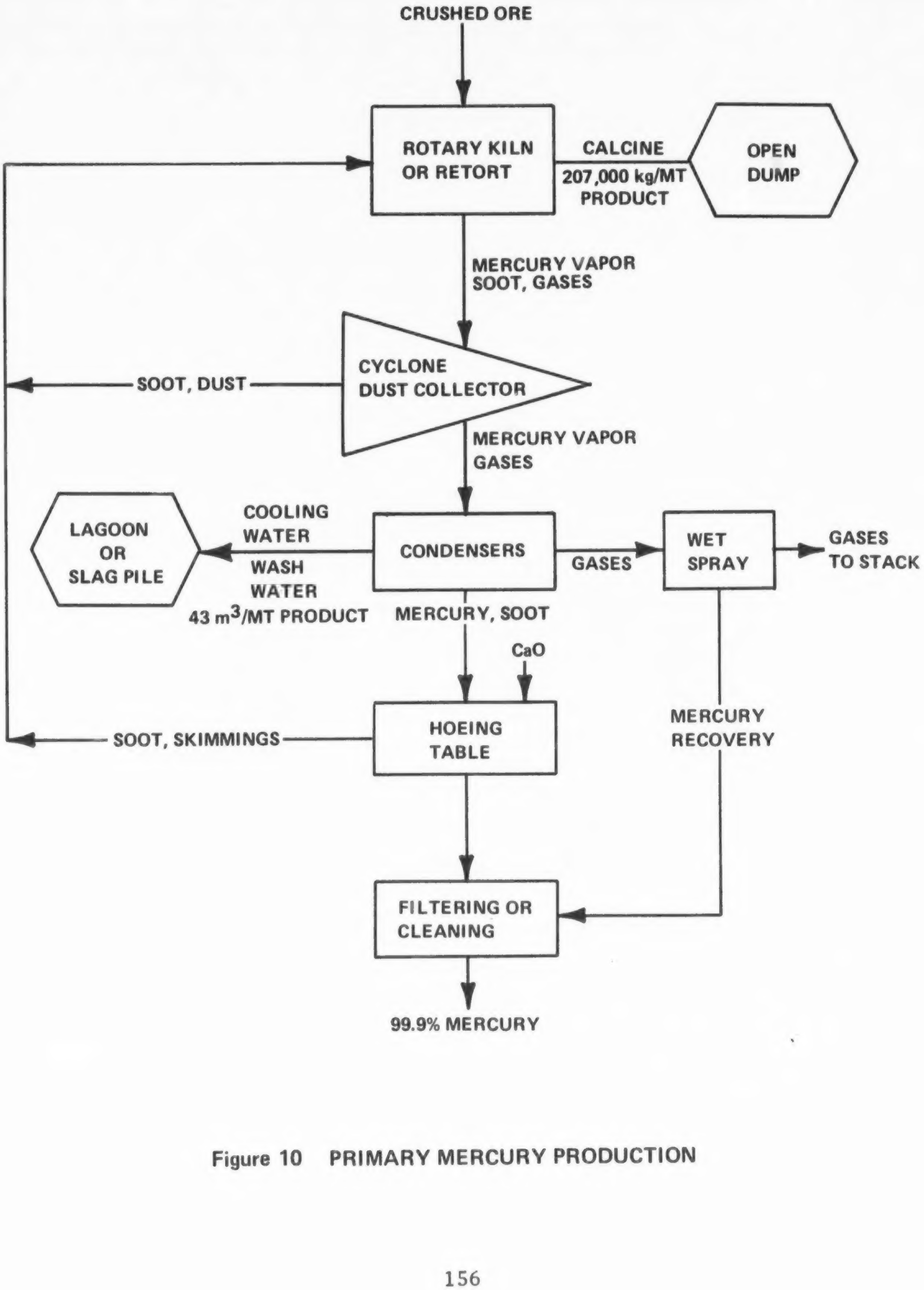

CRUSHED ORE
ROTARY KILN OR RETORT
CALCINE 207,000 kg/MT PRODUCT
OPEN DUMP
MERCURY VAPOR SOOT, GASES
CYCLONE DUST COLLECTOR
SOOT, DUST
MERCURY VAPOR GASES
LAGOON OR SLAG PILE
COOLING WATER
WASH WATER
43 m3/MT PRODUCT
CONDENSERS
GASES
WET SPRAY
GASES TO STACK
MERCURY, SOOT
CaO
HOEING TABLE
SOOT, SKIMMINGS
MERCURY RECOVERY
FILTERING OR CLEANING
99.9% MERCURY

Figure 10 PRIMARY MERCURY PRODUCTION

unlined lagoons is approximately 147,000 m^3. About 3,580 MT of calcine residue is open dumped per year from the typical plant. Two samples of calcine waste were obtained for chemical analyses. Results of analyses are given in Appendix A.

6.2.3 Waste Quantities

Table 67 summarizes the generation factors for each of the land disposed residuals in the primary mercury smelting and refining category (i.e. calcine residue condenser water). This table also gives the typical concentrations of potentially hazardous consitutents in a furnace calcine residue based mainly on the average analyses of two furnace calcine residues given in Appendix A. It is reported that mercury recovery from ore is about 95% using furnaces and 98% using retorts because of the higher efficiency of retorts. Calcine waste can vary widely in mercury content. A typical concentration of 200 ppm mercury in furnace or retort residue was used based on 98% recovery from a 1% ore.

Samples of solids from condenser wastewater and analyses thereof were not available, and therefore concentration factors could not be developed.

Using the waste generation factor and concentration factors for calcine residue contained in Table 67 estimates have been made for the quantity of land disposed residue and hazardous constituents thereof for a typical plant producing 500 flasks or 17.3 MT of mercury per year. These estimates are given in Table 68.

Table 69 gives state-by-state, regional and national quantities of land disposed residuals from mercury smelting based on production capacities and generation and concentration factors contained in Table 67. Estimates are given for 1974, 1977 and 1983. Estimates were based on the assumption that yearly national production by traditional methods will remain level at approximately 3,000 flasks through 1985 as forecast by U.S. Bureau of Mines.

6.3 Current Treatment and Disposal Practices

Calcine. Currently calcine from retorts or furnaces is open dumped. The greatest percentage of the primary mercury industry is located in arid or semi-arid regions. In addition insignificant leaching of trace metals was observed in solubility tests described in Appendix B. There appears to be little chance of groundwater contamination from calcine residue. Open dumping is considered environmentally adequate at this time and will not be discussed further.

Table 67

WASTE RESIDUAL FACTORS
(Primary Mercury Smelting and Refining)

Type of Waste	Generation Factor	Concentration Factor (ppm)									
		As	Pb	Cu	Zn	Ni	Hg	Mn	Sb	Cd	Cr
Calcine residue	207,000 ky/MT_n product	--	150	430	80	2600	200	1025	175	--	430
Condenser cooling and wash water*	43 m ^{3}/MT product	--	--	--	--	--	--	--	--	--	--

* Samples for analyses not available.

Table 68

YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL PRIMARY MERCURY SMELTER

Type of Waste	Total Waste Quartity (MT)	Quantity of Potentially Hazardous Constituents (MT)							
		As	Pb	Cu	Zn	Ni	Hg	Sb	Cr
calcine residue	17.3	--	0.54	1.54	0.29	9.31	0.72	0.63	1.54

*Quantities calculated from generation and concentration factors given in Table 67 based on annual production of 17 MT. Divide by 150 to obtain daily quantities. Multiply by 1.1 to convert to short tons.

Table 69
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY MERCURY PRODUCTION TOTAL CALCINE RESIDUE - 1974, 1977, 1983 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	DISPOSAL METHOD	CONSTITUENTS							
				Cr	Cu	Hg	Mn	Ni	Pb	Sb	Zn
CALIFORNIA	14,283	0	OPEN DUMP	6.14	6.14	2.85	14.3	37.1	2.14	2.50	1.14
NEVADA	7,142	↓	↓	3.07	3.07	1.43	7.1	18.5	1.07	1.25	0.57
EPA REGION IX	21,425	↓	↓	9.21	9.21	4.29	21.4	55.6	3.21	3.75	1.71
NATIONAL TOTAL	21,425	↓	↓	9.21	9.21	4.29	21.4	55.6	3.21	3.75	1.71

*THIS WASTE BELIEVED NONHAZARDOUS BASED ON CALSPAN SOLUBILITY TESTS DESCRIBED IN APPENDIX B. MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Condenser Water. Wastewater from condenser cooling or washdown is either sent to unlined lagoons or spread in the calcine dump. This waste is considered potentially hazardous due to the suspected presence of free mercury. Levels of technology for this wastewater are discussed in the following sections.

6.3.2 Present Treatment and Disposal Technology (Level I)

The practice of disposing of condenser cooling and wash water in unlined lagoons or in calcine dumps is not considered environmentally adequate.

6.3.3 Best Technology Currently Employed (Level II)

Level II treatment and disposal technology and adequacy thereof are the same as Level I.

6.3.4 Technology To Provide Adequate Health and Environmental Protection (Level III)

Lined lagoons should be used to store condenser wastewater and any other process wastewater.

Table 70 summarizes features of Levels I, II and III treatment and disposal technologies including assessment of the hazardous nature of residuals and the methods and adequacy of treatment and disposal methods.

6.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical mercury furnaces or retorting operations is considered in this section. The typical plant is assumed to produce 500 flasks (17.3 MT) of metallic mercury per year and operates 150 days/year.

6.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Condenser cooling and washdown water which may contain traces of metallic mercury flows into a lagoon. The flow is at a daily rate of The lagoon is designed to hold 20 years of effluent. Its characteristics are:

Volume	2,000 m^3	Depth of Excavation	0.95 m
Bottom Width	19.4 m	Dike Volume	858 m^3
Top Width	27.4 m	Dike Surface	2,426 m^2
Bottom Length	38.8 m	Total Width	37.6 m
Top Length	46.8 m	Total Length	57.0 m
Circumference	160.4 m	Required Area	0.21 ha

Table 70.

Treatment and Disposal Technology Levels

Smelting and Refining Category PRIMARY MERCURY SIC #3339

CONDENSER WASTEWATER

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Dilute slurry containing soot, traces of metallic mercury	Same as Level I	Same as Level I
Amount of Waste (m^3/MT Product)	43 m^3	43 m^3	43 m^3
Factors Affecting Hazardousness	May contain traces of metallic mercury and other heavy metals	Same as Level I	Same as Level I
Treatment/ Disposal Technology	Unlined lagoon or spread on calcine dump	Unlined lagoon	Lined Lagoon
Estimate of # + % of Plants Using Technology	6 - 10 (100%)	Not known	0

Table 70-(cont'd.)

CONDENSER WASTEWATER (continued)

Factor	Level 1	Level II	Level III
Adequacy of Technology	Inadequate	Inadequate	Adequate
Problems and Comments	Water tables probably deep - semi-arid climate	-	High cost per MT product
Non-Land Environmental Impact	Possible contamination of ground or surface water if percolation occurs	Same as Level I	None - percolation precluded
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Surveillance sump
Energy Requirements	Negligible	Negligible	Negligible

Practically all mercury plants are located in arid regions. The sizing of the lagoon assumes a net annual evaporation rate of about 63 cm/m^2 of surface area.

Capital and operating costs are shown in Table 71.

6.4.2 Cost of Best Technology Currently Employed (Level II)

The technology and costs of Level II technology are the same as for Level I.

6.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection (Level III)

The lagoon is lined. Associated capital and annual costs are shown below.

Capital Cost	Slurry
Lagoon Liner	$ 7,375
Total	$ 7,375

Annual Costs	
Construction Amortization	$ 855
Construction Repair and Maintenance	220
Insurance	75
Total	$ 1,150

Costs of Levels I, II and III treatment and disposal technologies for condenser wastewater slurries are summarized in Table 72. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of mercury product. Costs for each level of technology are also expressed as a percent of metal selling price.

Total industry costs for Levels I, II and III treatment and disposal technologies have been estimated in 1973 dollars and are presented in Table 72. The annualized industry cost for Level I and II treatment and disposal technology is 40,000 or 0.7% of estimated 1973 national sales. The annualized cost of Level III technology for the industry is $210,000 as a result of the need for lined lagoons for adequate environmental protection. This represents 1.5% of estimated 1973 national sales value.

Table 71

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY MERCURY PLANT

Capital Cost

	Slurry
Lagoon	
Site Preparation	
Survey	$ 130
Test Drilling	490
Sample Testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	1,140
Compacting	1,585
Fine Grading	1,090
Soil Poisoning	200
Transverse Drain Fields	345
Land	370
TOTAL	$ 6,800

Annual Cost

Land	$ 35
Amortization	
Construction	745
Equipment	--
Operating Personnel	--
Repair and Maintenance	
Construction	130
Equipment	--
Energy	--
Fuel	
Electricity	
Taxes	10
Insurance	70
TOTAL	$ 990

Table 72

COST SUMMARY FOR PRIMARY MERCURY SMELTING AND REFINING

Annual Production:	Model Plant 17.3 MT	Industry 74.0 MT

Waste (type)	Amount (MT/MT of production)
Slurry	44

Cumulative Unit Waste Disposal Costs:

	Level I		Level II		Level III	
Waste (type)	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.
Slurry	Wet		Wet		Wet	
Capital Cost	8.93	393.06	8.93	393.06	18.62	819.36
Annual Cost	1.30	57.25	1.30	57.25	2.81	123.69
Total						
Capital Cost		$393.06		$393.06		$819.36
Annual Cost		57.25		57.25		123.69

Cumulative Industry Waste Disposal Costs ($ million):

	Level I		Level II		Level III	
Waste (type)	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slurry	$0.03	$0.004	$0.03	$0.004	$0.06	$0.009
Total	$0.03	$0.004	$0.03	$0.004	$0.06	$0.009

Metal Price: $8,396.08/MT

Treatment Cost as a Percent of Metal Selling Price:

	Level I		Level II		Level III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slurry	4.7%	0.7%	4.7%	0.7%	9.8%	1.5%
Total	4.7%	0.7%	4.7%	0.7%	9.8%	1.5%

7.0 PRIMARY SMELTING AND REFINING OF TITANIUM (SIC 3339)

7.1 INDUSTRY CHARACTERIZATION

There are only two primary titanium, (i.e., titanium sponge) producers in the U.S. One is Reactive Metals, Niles, Ohio and the other is Titanium Metals Corporation, Henderson, Nevada (jointly owned by National Lead and Allegheny Ludlum). Total U.S. plant capacity is approximately 15,230 MT of titanium sponge metal. State, regional and national distribution of titanium sponge metal plants is given in Table 73. Approximately 9 plants in the United States take the sponge metal and purify it to ingots in electric furnaces. There are virtually no wastes associated with this operation. For this reason this section of the report is restricted to sponge metal production. Domestic production of titanium sponge is not published. The selling price of sponge in 1973 was given as $1.42/lb. Assuming that the above two plants were operating at capacity, national sales are estimated as approximately $47,800,000.

Production of titanium sponge consists of chlorination of rutile (TiO_2) concentrates to produce titanium chloride gas followed by reduction to titanium sponge.

7.2 WASTE CHARACTERIZATION

The typical plant produces approximately 7600 MT of titanium sponge per year and operates 350 days per year. This is the average production of the two plants.

7.2.1 Process Description

Feed to the plant is estimated at 13,670 MT/yr of rutile (TiO_2) concentrate containing 94% TiO_2. The concentrates are treated (See Figure 11) with chlorine gas to produce $TiCl_4$ gas which is then condensed, reduced with magnesium metal and purified with aqua regia to produce titanium sponge. The titanium sponge is then melted and purified in an electric furnace to produce pure titanium ingot. Magnesium chloride from the reduction process is electrolyzed to produce chlorine gas and magnesium metal which are recycled as shown in Figure 11.

7.2.2 Description of Waste Streams

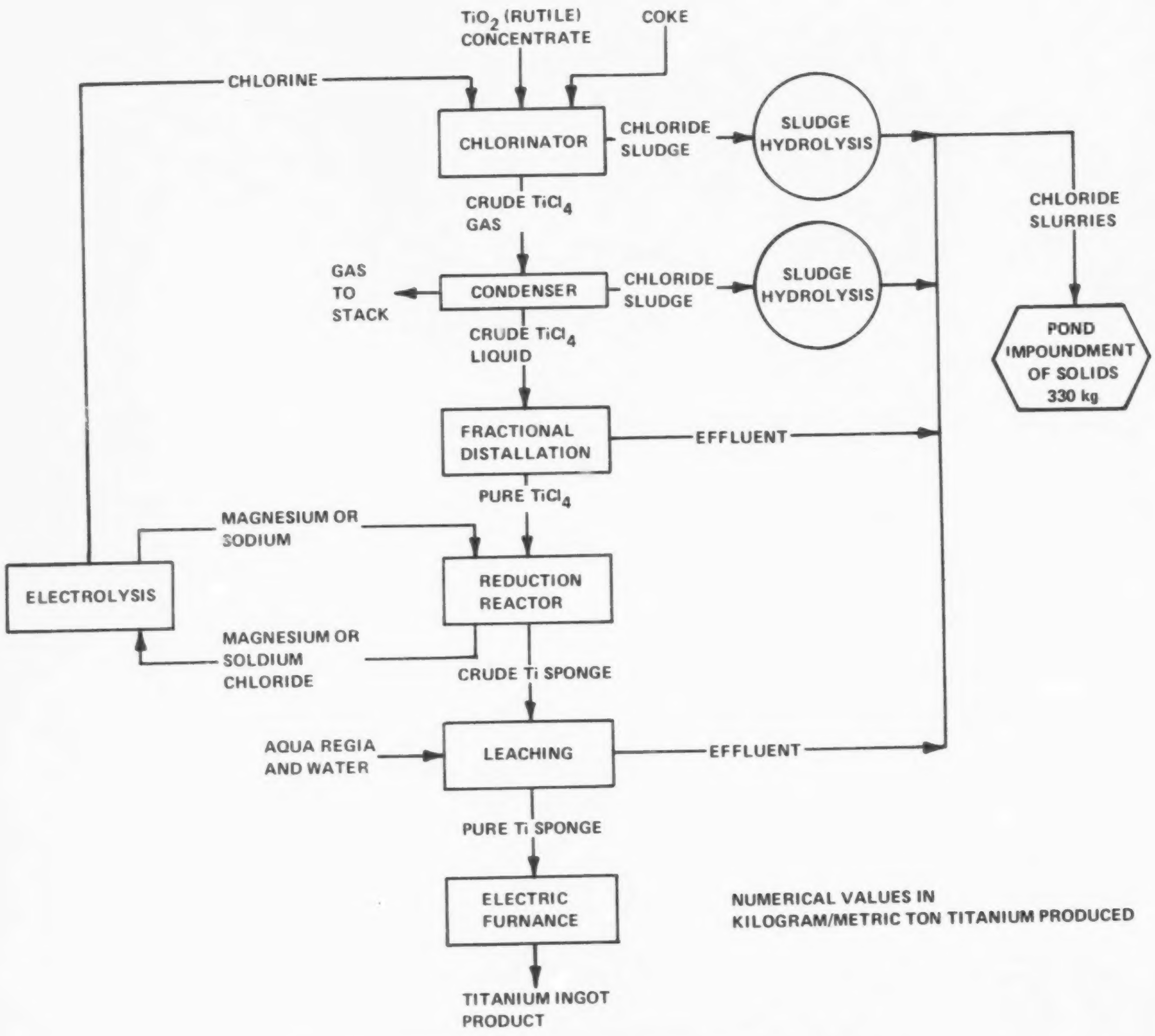
Chlorinator and Condenser Sludges. Impurities in the rutile feed are mainly aluminum, columbium, iron, silicon, vanadium, and zirconium. These elements plus some of the carbon, chlorine, and titanium present in the chlorination mix and the condenser form sludges which are settled in a pond. Sponge melting is carried out in a vacuum furnace which produces no slag. Much of the chlorine used in chlorinating the rutile, and the sodium or magnesium used in reducing $TiCl_4$ to metallic titanium metal are recovered by electrolysis and recycled in the process.

TABLE 73

GEOGRAPHICAL DISTRIBUTION OF PRIMARY TITANIUM SPONGE METAL PRODUCTION CAPACITY

State	No. Of Plants	Plant Distribution by Capacity (MT)		Plant Distribution by Process	Capacity Per Process (MT)
		5680	9550	CR*	CR*
Ohio	1	1		1	5680
Nevada	1		1	1	9550
EPA Region					
V	1	1		1	5680
IX	1		1	1	9550
National	2			2	15,230

* CR - Chlorination, reduction


TiO$_2$ (RUTILE) CONCENTRATE
COKE
CHLORINE
CHLORINATOR
CHLORIDE SLUDGE
SLUDGE HYDROLYSIS
CHLORIDE SLURRIES
CRUDE TiCl$_4$ GAS
GAS TO STACK
CONDENSER
CHLORIDE SLUDGE
SLUDGE HYDROLYSIS
POND IMPOUNDMENT OF SOLIDS 330 kg
CRUDE TiCl$_4$ LIQUID
FRACTIONAL DISTALLATION
EFFLUENT
PURE TiCl$_4$
MAGNESIUM OR SODIUM
ELECTROLYSIS
REDUCTION REACTOR
MAGNESIUM OR SOLDIUM CHLORIDE
CRUDE Ti SPONGE
AQUA REGIA AND WATER
LEACHING
EFFLUENT
PURE Ti SPONGE
ELECTRIC FURNANCE
TITANIUM INGOT PRODUCT

NUMERICAL VALUES IN
KILOGRAM/METRIC TON TITANIUM PRODUCED

Figure 11 PRIMARY TITANIUM SPONGE PRODUCTION

Sludges from the chlorinator and the condenser are the only significant solid waste. As shown in Figure 11 these sludges, amounting to 330 Kg/MT of titanium sponge (dry weight) or 2508 MT/yr, along with a small quantity of distillation effluent and spent acid from leaching of titanium sponge is sent to impoundment ponds, which are dredged frequently. The amount of combined sludge dredged from the settling pond is estimated at 2520 MT/yr or 7.2 MT/day (dry basis). This sludge is trucked to a landfill site located within a few miles of the plant by a contractor. It has been stated that chlorinator and condenser waste sludge is about 40 percent water soluble and that the water soluble portion contains potentially hazardous chloride and chloride-oxide complexes of chromium, titanium, vanadium and other heavy metals (Ref. 4). In addition, HCl gas may be released to the atmosphere in an anaerobic environment. These sludges are therefore considered potentially hazardous.

7.2.3 Waste Quantities

Table 74 gives the generation factors for the chlorination-condenser sludges which are lagoon disposed. This table also gives the typical concentrations of potentially hazardous constituents. Concentrations of potentially hazardous constituents were obtained from chlorinator sludge analyses contained in Bureau of Mines Report of Investigations 7221 (Ref. 4). Average analyses of 8 samples showed the following percentages of chlorine and various metals:

Cl	18.7
Cr_2O_3	1.7
TiO_2	17.4
V_2O_5	4.6
ZrO_2	4.7

Chloride and titanium are relatively non-toxic compared to many other substances. However their very high concentrations and high solubility present definite groundwater pollution hazards.

Table 75 gives the yearly generated quantity of sludge and potentially hazardous constituents thereof for a typical plant producing 7600 MT of titanium sponge operating 350 days per year. Generation and concentration factors contained in Table 74 were used in calculations for Table 75.

Table 76 gives estimates for state-by-state, regional and national sludge generation and hazardous constituents for 1974, 1977, and 1983. The only two states generating hazardous waste from titanium sponge production are Ohio and Nevada with Nevada generating twice as much as a

TABLE 74 WASTE RESIDUAL FACTORS - PRIMARY TITANIUM REFINING

Type of Waste	Generation Factor	Concentration Factor (ppm)				
		V	Cr	Zr	Ti	Cl
Chlorinator and condenser sludges	330 kg/MT of sponge titanium metal	25,780	11,630	34,770	104,400	187,000

TABLE 75. YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL TITANIUM REFINING PLANT*

Type of Waste	Total Waste Quantity (MT)	Quantity of Potentially Hazardous Constituents (MT)				
		V	Cr	Zr	Ti	Cl
Chlorinator and condenser sludges	2508	64.7	29.2	87.2	261.8	469

* Quantities calculated from generation and concentration factors given in Table 74 based on annual production of 7600 MT. Divide by 350 to obtain daily quantities, multipy by 1.1 to convert to short tons.

Table 76
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY TITANIUM PRODUCTION
TOTAL CHLORINATOR AND CONDENSER SLUDGE - 1974, 1977, 1983 (METRIC TONS)*

STATE	TOTAL LAND DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS			
					Cl	Cr	Ti	V
OHIO	1900	1900	810	LAGOON	360	32	330	87
NEVADA	3200	3200	1360	LANDFILL	600	54	560	150
EPA REGION V	1900	1900	810	LAGOON	360	32	330	87
IX	3200	3200	1360	LANDFILL	600	54	560	150
NATIONAL TOTAL	5100	5100	2170		960	86	890	240

*MULTIPLY BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

result of larger plant capacity. There is no basis for estimating plant capacity changes or changes in pollution control technology which would change the quantities of land disposed waste in 1977 and 1983 as compared to 1974 since production quantities of sponge metal from the two domestic plants are not published.

7.3 TREATMENT AND DISPOSAL TECHNOLOGY

7.3.1 Current Treatment and Disposal Practices

The only significant solid waste is the chlorinator-condenser sludge. At the plant located in EPA Region IX, sludge is reacted with water and settled in a pond. The settled sludge is clammed out and hauled to a commercial landfill site. Daily cover of waste is practiced at this site. At the plant located in Region V sludge is trucked by a contract disposer to large lagoons constructed in highly impermeable glacial till and clay underlain by shale. Since this waste is considered potentially hazardous it is discussed in more detail in the following sections.

7.3.2 Present Treatment and Disposal Technology

As discussed previously chlorinator-condenser sludge is disposed of either in a commercial landfill or a industrial sludge lagoon. This practice is inadequate if toxic constituents (i.e. chromium, vanadium, others) are found to percolate through soils to underlying aquifers.

7.3.3 Best Technology Currently Employed (Level II)

At the present time, chlorination condenser sludge containing minor amounts of solids from distillation and leaching is either put into a daily covered landfill or into sludge lagoons. Either of these practices will qualify as Level II technology. If significant movement of the chloride salts of chromium, titanium, vanadium and others into water tables occurs, then these practices could not be considered environmentally adequate.

7.3.4 Technology to Provide Adequate Health and Environmental Protection (Level III)

As discussed previously, it has been shown that chlorinator waste sludge is highly water soluble and the water fraction contains chloride salts and complexes of chromium, vanadium, titanium and other heavy metals. For this reason, lined lagoons should be used for ultimate disposal of these sludges.

Features of Levels I, II and III treatment and disposal technologies including assessment of the hazardous nature of residuals and the methods and adequacy of treatment and disposal methods are summarized in Table 77.

Table 77

Treatment and Disposal Technology Levels

Smelting and Refining Category PRIMARY TITANIUM SIC 3339

CHLORINATOR SLUDGES

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Fine agglomerated particles containing oxides, silicates and chlorides	Same as I	Same as I
Amount of Waste (kg/MT Product)	330 kg	330 kg	330 kg
Factors Affecting Hazardousness	Presence of water soluble chloride - oxide complexes of chrome, vanadium, zirconium; acid	Same as Level I	Same as Level I
Treatment/ Disposal Technology	Daily covered landfill or sludge lagoon	Same as Level I	Lined lagoons
Estimate of # + % of Plants Using Technology	Landfill - 1 (50%) Lagoon - 1 (50%)	Same as Level I	0

Table 77 -(cont'd.)

CHRLORINATOR SLUDGES (continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate	Same as Level I	Adequate
Problems and Comments	Demonstrated water solubility of 40%	-	-
Non-Land Environmental Impact	Possible contamination of grand and surface water if percolation of leachate occurs	Same as Level I	None - percolation precluded
Compatibility With Existing Facilities	Compatible	Compatible	May be difficult to line lagoons in use
Monitoring & Surveillance Methods	None	None	Monitoring sump
Energy Requirements	Negligible	Negligible	Negligible

7.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. The representative plant is assumed to have an annual production of 7,600 MT titanium sponge metal and operates 350 days per year.

7.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Land disposed solid waste consists of chlorinator sludge which accumulates in settling pits. The daily flow is 1135 m^3 containing 7.16 MT solids. The settling pits are dredged by an outside contractor and carried by tank truck to an off-site lagoon. An average of 27.5 m^3 of sludge are hauled daily.

Two settling pits, each designed to accommodate a 1 day flow are provided. Their dimensions are 30 x 8 x 5 m. The contractor tank trucks are assumed equipped with slurry pumps. It is estimated that 2 hrs/day of truck time are required for dredging and transporting the sludge.

The only capital costs incurred by the plant are the construction of the settling pits. Annual costs are based on the contractor's construction and operation costs. The latter are computed similarly as for on-site operations and then increased by a factor of 2 to account for transportation, general administrative expenses and profit. The outside contractor is assumed to utilize a lagoon with a volume of 192,500 m^3, sufficient to contain 20 years of sludge wastes.

The capital cost to the plant for excavating and forming the settling pits is $3,190. Other costs are shown in Table 78.

TABLE 78

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY

Primary titanium (Chlorinator and Condenser Sludges)

Capital Cost	$ 3,190
Annual Costs	
Construction Amortization	235
Insurance	30
Contractor Charges	45,560
Total	$49,015

7.4.2 Cost of Best Technology Currently Employed (Level II)

The technology and costs of Level II are the same as those for Level I.

7.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection (Level III)

The lagoon used for potentially hazardous chlorination sludge disposal is lined. This results in an additional annual cost of $89,500 to the plant.

Costs of Levels I, II and III Treatment and Disposal Technology are summarized in Table 79 for sludge residuals from primary titanium refining. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of titanium product. Costs for each level of technology are also expressed as a percent of metal selling price.

Total industry costs for Levels I, II and III treatment and disposal technolgoies have been estimated in 1973 dollars and are presented in Table 79. The annualized industry costs for Levels I and II treatment and disposal technology is estimated as $70,000 or 0.16% of estimated 1973 sales values. The annualized industry cost for Level III treatment and disposal technology (i.e. adequate for environmental protection) is estimated as $230,000 or 0.54% of estimated 1973 sales.

TABLE 79 COST SUMMARY FOR PRIMARY TITANIUM REFINING

Annual Production: Model Plant 7,600 MT Industry 13,636 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	1.7

Cum. Unit Waste Disposal Costs:

Waste (Type)	Level								
	I			II			III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$0.26	$0.10	$0.42	$0.26	$0.10	$0.42	$0.26	$0.10	$0.42
Annual Cost	3.01	1.20	4.95	3.01	1.20	4.95	10.17	4.07	16.73
Total Capital Cost	-		0.42	-		0.42	-		0.42
Total Annual Cost	-		4.95	-		4.95	-		16.73

Cum. Industry Waste Disposal Costs ($ Million)

Waste (Type)	Level					
	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$0.01	$0.07	$0.01	$0.07	$0.01	$0.23
Total	0.01	0.07	0.01	0.07	0.01	0.23

1973 Metal Price: $3,124/MT

Treatment Cost as a Percent of Metal Selling Price

Waste (Type)	Level					
	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.01%	0.16%	0.01%	0.16%	0.01%	0.54%
Total	0.01	0.16	0.01	0.16	0.01	0.54

8.0 PRIMARY REFINING OF TUNGSTEN (SIC 3339)

8.1 INDUSTRY CHARACTERIZATION

The tungsten producing and processing industry is small and rather specialized compared with major nonferrous metals industries. Production of metal powder was 3,404 MT in 1971, 4,322 MT in 1972 and 5,634 MT in 1973 (Ref. 2). Geographical distribution of production capacity by state and EPA region is given in Table 80. The selling price of tungsten metal powder varied from $4.50 to $6.70 per pound. Using a price of $5.00 the national sales are estimated as $61,974,000 for 1973.

8.2 WASTE CHARACTERIZATION

From the limited information available, it is estimated that the average production of tungsten metal at the 15 plants in the United States which produce metal or the final precursor to metal (i.e., ammonium paratungstate) is 300 MT/year. Assuming 350 days/year operation, daily capacity is 0.86 MT. It is believed that wet (hydrometallurgical) processing is universal in the industry, but with very substantial variations in process details.

8.2.1 Process Description

A flow diagram showing tungsten production processes and sources of land disposed waste is shown in Figure 12. Ore concentrates containing 60-70 percent tungsten expressed as WO_3 are treated with hot HCl to solubilize tungsten and precipitate impure tungstic acid. The crude tungstic acid is leached with sodium hydroxide to produce a solution of sodium tungstate which can be further purified by crystallization, re-dissolution, re-precipitation, final digestion in HCl, and reaction with ammonia to form pure ammonium paratungstate (APT). The crystallized APT is then reduced pyrometallurgically with hydrogen gas to form pure tungsten metal powder.

8.2.2 Description of Waste Streams

Digestion Residue As shown in Figure 12, a solid digestion residue (Filter cake) remaining from the initial digestion of the ore constitutes the primary residue from the tungsten production process. This material in the amount of approximately 15 MT/year (50 Kg per metric ton of tungsten product) is stored on land (for many years) at some plants and is stored in drums at other plants. It apparently contains sufficient metal values to warrant eventual re-treatment. Analysis of a sampled digestion residue is contained in Appendix A. Copper is present in significant concentration (3.8%) and traces of zinc and lead are also present. In solubility tests as described in Appendix B copper, nickel and zinc were found to leach significantly (90, 60, and 1.5 ppm respectively). Digestion residue is therefore considered potentially hazardous.

Table 80
GEOGRAPHICAL DISTRIBUTION OF PRIMARY TUNGSTEN PRODUCTION CAPACITY

STATE	NO. OF PLANTS	PLANT DISTRIBUTION BY CAPACITY(MT)		PLANT DISTRIBUTION BY PROCESS	CAPACITY PER PROCESS (MT)
		100-500K	600-900K	HR	HR
CALIFORNIA	1	0	1	1	900
ILLINOIS	1	1	0	1	220
NEW JERSEY	2	2	0	2	450
NEW YORK	2	2	0	2	680
NEVADA	2	2	0	2	220
OHIO	1	0	1	1	900
OREGON	1	1	0	1	220
PENN.	3	3	0	3	900
TEXAS	2	2	0	2	200
EPA REGION					
II	4	0	0	4	1130
III	3	0	0	3	900
V	2	0	0	2	1120
VI	2	0	0	2	200
IX	3	0	0	3	1120
X	1	0	0	1	220
NATION	15	0	0	15	4690

HR - HYDROMETALLURGICAL FOLLOWED BY REDUCTION

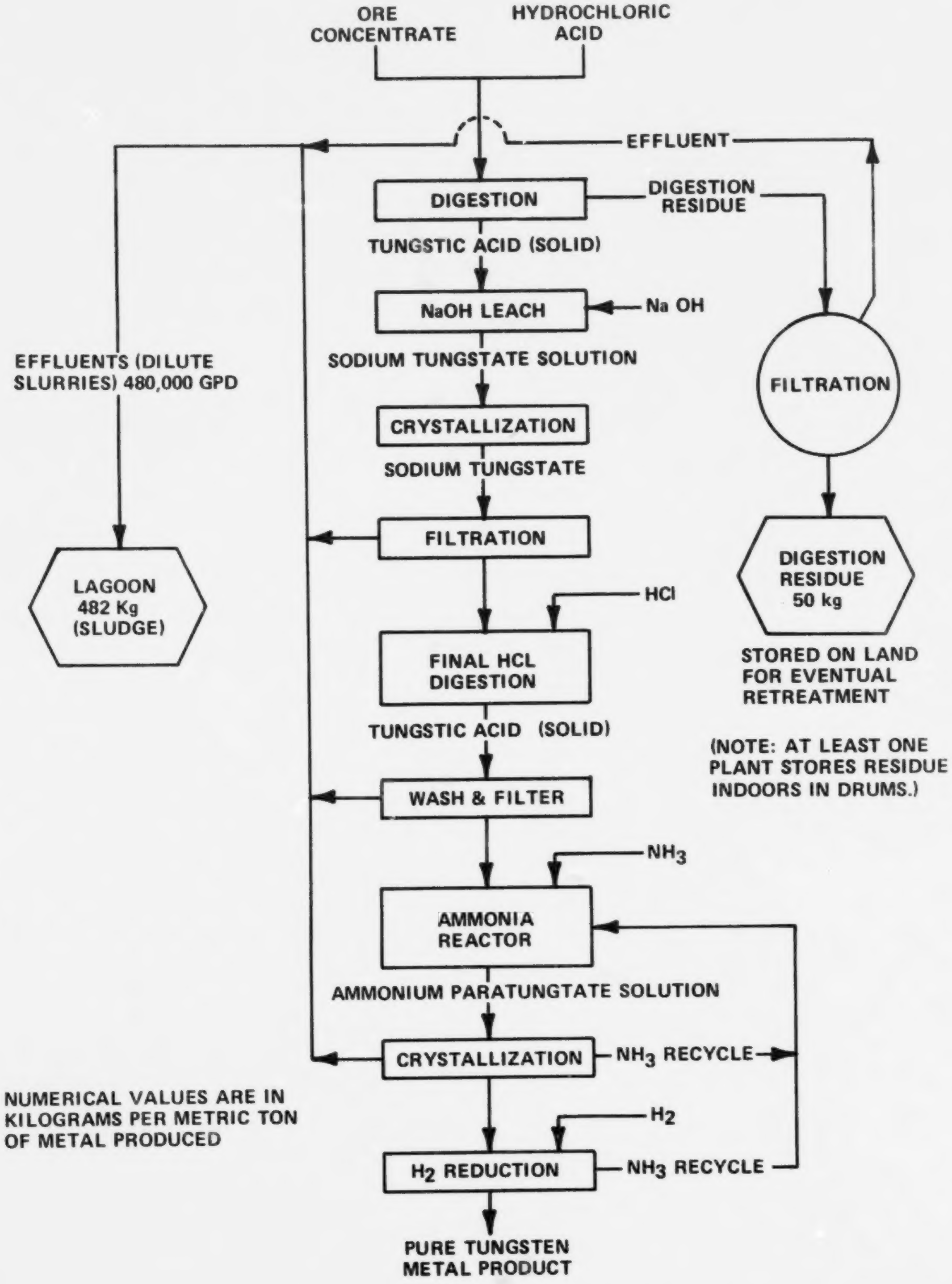

ORE CONCENTRATE
HYDROCHLORIC ACID
EFFLUENT
DIGESTION
DIGESTION RESIDUE
TUNGSTIC ACID (SOLID)
NaOH LEACH
Na OH
SODIUM TUNGSTATE SOLUTION
EFFLUENTS (DILUTE SLURRIES) 480,000 GPD
FILTRATION
CRYSTALLIZATION
SODIUM TUNGSTATE
FILTRATION
LAGOON 482 Kg (SLUDGE)
DIGESTION RESIDUE 50 kg
HCl
FINAL HCL DIGESTION
STORED ON LAND FOR EVENTUAL RETREATMENT
TUNGSTIC ACID (SOLID)
(NOTE: AT LEAST ONE PLANT STORES RESIDUE INDOORS IN DRUMS.)
WASH & FILTER
NH_3
AMMONIA REACTOR
AMMONIUM PARATUNGTATE SOLUTION
CRYSTALLIZATION
NH_3 RECYCLE
NUMERICAL VALUES ARE IN KILOGRAMS PER METRIC TON OF METAL PRODUCED
H_2
H_2 REDUCTION
NH_3 RECYCLE
PURE TUNGSTEN METAL PRODUCT

Figure 12 PRIMARY TUNGSTEN REDUCTION

Wastewaters. Dilute slurries from each process containing dissolved and suspended solids flow into a lagoon. Daily flow to the lagoon is 1817 m^3 (480,000 gallons) and carries approximately 395 kg of solids (482 Kg/MT tunsten product). Yearly accumulation of wet sludge in the lagoon is approximately 345 MT containing 138 MT of solids. About once a year sludge is dredged from the lagoon and dumped on land.

Samples of this sludge were not obtained but consideration of the composition of ore concentrate and processing reagents indicate that soluble salts of sodium and ammonium will be present in addition to traces of arsenic, copper, lead and zinc. The fine particulate nature of wastewater sludges and solubility of sodium salts indicates that significant leaching could occur. This waste is therefore considered potentially hazardous at this time.

8.2.3 Waste Quantities

Table 81 gives waste generation factors for a primary Tungsten plant and Table 82 gives the yearly generation of wastewater sludge and digestion residue from a typical plant.

Based on waste factors for digestion residue and wastewater as previously given, state by state production estimates, and material balance calculations, estimates of land disposed wastes and potentially hazardous constituents thereof were made. These estimates are given on state by state, regional and national levels in Table 83 for 1974, 1977 and 1983. Production and pollution control technology in 1977 and 1983 are assumed to not change significantly, thereby producing about the same quantities of land disposed wastes as in 1974.

8.3 TREATMENT AND DISPOSAL TECHNOLOGY

8.3.1 Current Waste Treatment and Disposal Practices

Digestion residue is usually stored on land before recycling. Wastewater from filtration processes is settled in unlined lagoons, dredged once a year and open dumped on land. As discussed previously both of these wastes are considered potentially hazardous. Levels of technology for their treatment and disposal are discussed in the following sections.

8.3.2 Present Treatment and Disposal Technology (Level I)

Digestion Residue The digestion residue remaining after treatment of tungsten ore concentrate is stored on land for a period of one or more years before eventual recycling. Storage on land constitutes Level I practice. This is environmentally inadequate if trace metals (Cu, Ni, Zn, others) percolate through permeable soils to groundwater.

TABLE 81.

WASTE RESIDUAL FACTORS, TUNGSTEN

Type of Waste	Generation Factor Kg/MT	Concentration Factor (ppm) As	Pb	Cu	Zn
Sludge	482	3.5	137	137	26
Digestion Residue	50	-	1.0	377	8.9

TABLE 82. YEARLY GENERATION OF WASTE RESIDUALS FROM TYPICAL TUNGSTEN PLANT

Type of Waste	Total Waste Quantity Quantity, (MT)	Quantity of Potentially Hazardous Constituents (MT)			
		As	Pb	Cu	Zn
Sludge	145	0.051	1.98	1.98	0.38
Digestion Residue	15				

* Quantities calculated from generation and concentration factors given in Table 81 based on annual production of 300 MT. Divide by 350 to obtain daily quantities, multiply by 1.1 to convert to short tons.

Table 83
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY TUNGSTEN PRODUCTION - 1974, 1977, 1983, DRY WEIGHTS*

STATE	SLUDGE								DIGESTION RESIDUE						
	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS				TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS		
					As	Cu	Pb	Zn					Cu	Pb	Zn
NEW JERSEY	217	217	23	LAGOON	1.0	10	10	2.0	NOTE(1)	23	0.92	STORED ON LAND	0.9	0.002	0.02
NEW YORK	328	328	35		1.5	15	15	3.0		34	1.33		1.3	0.003	0.03
PENNSYLVANIA	434	434	46		2.0	20	20	4.0		45	1.74		1.7	0.004	0.04
OHIO	434	434	46		2.0	20	20	4.0		45	1.74		1.7	0.004	0.04
ILLINOIS	106	106	12		0.5	5	5	1.0		11	0.41		0.4	0.001	0.01
TEXAS	97	97	12		0.5	5	5	1.0		11	0.41		0.4	0.001	0.01
CALIFORNIA	434	434	46		2.0	20	20	4.0		45	1.74		1.7	0.004	0.04
NEVADA	106	106	12		0.5	5	5	1.0		11	0.41		0.4	0.001	0.01
OREGON	106	106	12		0.5	5	5	1.0		11	0.41		0.4	0.001	0.01
EPA REGION II	545	545	58		2.5	25	25	5.0		57	2.25		2.2	0.005	0.05
REGION III	434	434	46		2.0	20	20	4.0		45	1.74		1.7	0.004	0.04
REGION V	540	540	58		2.5	25	25	5.0		56	2.15		2.1	0.005	0.05
REGION VI	97	97	12		0.5	5	5	1.0		11	0.41		0.4	0.001	0.01
REGION IX	540	540	58		2.5	25	25	5.0		56	2.15		2.1	0.005	0.05
REGION X	106	106	12		0.5	5	5	1.0		11	0.41		0.4	0.001	0.01
NATIONAL TOTAL	2262	2262	244		10.5	105	105	21.0		236	9.11		8.9	0.021	0.21

*MULTIPLY SLUDGE QUANTITIES BY 2.5 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Wastewater The wastewater from various filtration and washing processes as shown in Figure 12 is sent to unlined lagoons. Solids from this water produce sludge at a rate of 482 Kg/MT of tungsten metal produced. The wet sludge amounting to 345 MT (138 MT dry weight) per year is dredged from the pond once a year and open dumped on land. This practice is inadequate if heavy metals leach and percolate through permeable soils to groundwater.

8.3.3 Best Technology Currently Employed (Level II)

Digestion Residue At least one plant is storing digestion residue in drums rather than on land. Since this practice precludes any leaching of potentially hazardous constituents, it constitutes Level II practice.

Wastewater Sludge At least one company is chemically fixing lagoon sludges before disposing on land. Since this practice will preclude or greatly inhibit leaching of potentially hazardous constituents, it is considered a Level II practice and environmentally adequate.

8.3.4 Technology to Provide Adequate Health and Environmental Protection (Level III)

Digestion Residue Storage of digestion residues in drums or equivalent storage will preclude any leaching thereby constituting Level III practice. This is the same as Level II technology.

Wastewater Sludge If significant leaching of hazardous constituents from wastewater sludge is demonstrated then the use of lined lagoons will be needed. Chemical fixation of sludge removed from lagoons before land disposal will be Level III practice if leaching of hazardous constituents is demonstrated.

Features of Levels I, II and III treatment and disposal technology are summarized in Tables 84 and 85.

8.4 COST ANALYSIS

In the last section various treatment and disposal technologies currently employed, or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. The typical plant produces 300 MT of tungsten annually and operates 350 days per year.

8.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Tungsten digestion residue is accumulated at a rate of 15 MT/year. It is stored on land for months to years before eventual retreatment. Hauling and storage costs are insignificant.

Various filtration wastewaters are sent to a lagoon. Daily inflow is about 1,800 m^3. Three hundred and forty-five MT of wet sludge is formed containing 138 MT of solids.

Table 84

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary Tungsten SIC 3339

DIGESTION RESIDUE

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Cake of silt and sand size particles; acid with chlorides	Same as I	Same as I
Amount of Waste (kg/MT Product)	50 kg	50 kg	50 kg
Factors Affecting Hazardousness	Contains heavy metals, including As, Cu, Pb, Zn	Same as I	Same as I
Treatment/ Disposal Technology	Land Storage	Stored in Drums	Stored in Drums
Estimate of # + % of Plants Using Technology	14 (93%)	1 (7%)	1 (7%)

Table 84-(cont'd.)
DIGESTION RESIDUE - continued

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if residue is leached	Adequate	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	Possible contamination of ground water or surface water	None	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	Monitoring wells;	Check containers for corrosion and leakage	Same as II
Energy Requirements	Negligible	Negligible	Negligible

Table 85

Treatment and Disposal Technology Levels

Smelting and Refining Category Primary Tungsten SIC 3339

Sludge from filtration wastewaters

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Aggregates of colloidal silt and sand size particles; chlorides, sodium	Same as I	Same as I
Amount of Waste (M^3/MT Product)	11,800	$13m^3$	$13m^3$
Factors Affecting Hazardousness	Contains heavy metals including As Cu, Pb, Zn, soluble chloride and ammonium salts	Same as	Same as I
Treatment/ Disposal Technology	Unlined lagoons for sludge settling; ultimate disposal in open dump	Unlined settling lagoons and chemical fixation of sludge before open dumping	Lined settling lagoons and chemical fixation of sludge before open dumping
Estimate of # + % of Plants Using Technology	14 (93%)	1 (7%)	0

Table 85-(cont'd.)

SLUDGE - continued

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if sludge is leached	Inadequate if leaching from lagoon	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	Possible contamination of ground water or surface water	Same as I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible but may be difficult to line lagoons in use
Monitoring & Surveillance Methods	None	None	Monitoring sump
Energy Requirements	Negligible	Negligible	Negligible

The settling pond characteristics needed to accommodate one year input of sludge with adequate retention time are:

Volume	7,500 m^3
Bottom Width	31 m
Top Width	43 m
Bottom Length	62 m
Top Length	74 m
Total Depth	3 m
Depth of Excavation	1.25 m
Circumference	245 m
Dike Volume	2,783 m^3
Dike Surface	3,308 m^2
Total Width	56 m
Total Length	87 m
Required Area	.5 ha

The pond is dredged once a year at which time 345 MT of wet sludge are removed, deposited adjacent to the pond, allowed to dry and then hauled to an on-site dump. About 14 hrs/yr of dragline time are required for dredging and 16 hours of frontloader and truck time to haul the sludge to the dump.

The dump extends over a .5 ha area, which is sufficient to contain 20 years of sludge waste. Forty hours/yr of bulldozer time are assigned to the dump for grading and compacting.

Level I capital and annual costs are shown in Table 86.

8.4.2 Cost of Best Technology Currently Employed (Level II)

The digestion residue is stored in 0.2 m^3 (55 gal) drums for a year before being reprocesses. The sludge from filtration wastewaters is chemically fixed before land disposal. Associated capital and annual costs are given in Table 87.

8.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection (Level III)

Level III technology is the same as Level II except that the lagoon is lined. Additional costs incurred are:

Capital Cost	
Lagoon Liner	\$16,045
Total	\$16,045

Table 86

Cost of Level I Treatment and Disposal Technology
Primary Tungsten plant

Capital Cost

	Sludge
Lagoon	
Site Preparation	
Survey	$ 315
Test Drilling	490
Sample Testing	250
Report Preparation	1,200
Construction	
Excavation & Forming	3,700
Compacting	5,150
Fine Grading	1,490
Soil Poisoning	305
Transverse Drain Fields	515
Land	880
Equipment	
Dump Truck	500
Frontloader	400
Dragline	490
Bulldozer	100
Dump	
Survey	375
Land	1,055
Total	$16,895

Annual Cost

Land	$ 195
Amortization	
Construction	1,600
Equipment	1,279
Operating Personnel	1,082
Repair and Maintenance	
Construction	335
Equipment	100
Energy	
Fuel	130
Electricity	130
Taxes	50
Insurance	150
	$ 5,001

TABLE 87

Cost of Level II treatment and Disposal Technology
Primary Tungsten Plant

Capital Cost

	Sludge	Digestion Residue
Equipment		
Storage Drums		$1,100
Total	-	$1,100

Annual Cost

	Sludge	Digestion Residue
Land	-	-
Amortization		
Construction	-	-
Equipment	-	$ 175
Operating Personnel	-	-
Repair and Maintenance		
Construction	-	-
Equipment	-	55
Energy		
Fuel	-	-
Electricity	-	-
Taxes	-	-
Insurance	-	10
Chem. Fix.	$ 374	-
Total	$ 374	$ 240

Annual Cost	
Construction Amortization	$ 1,865
Construction Maintenance & Repair	480
Insurance	160
Total	$ 2,505

Costs of Levels I, II and III treatment and disposal technologies are summarized in Table 88 for sludge and digestion residuals from primary tungsten refining. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of tungsten product. Costs for each waste at each level of technology are also expressed as a percent of metal selling price.

Total industry costs for Levels I, II and III treatment and disposal technologies have been estimated in 1973 dollars and are presented in Table 88. The annualized industry costs for Level I treatment and disposal technology of sludge and digestion residue is estimated as $90,000 for 1973 or 0.13% of estimated national sales. The estimated 1973 annualized industry cost is $420,000 or 0.57% of estimated national sales for Level II technology. The estimated 1973 annualized industry cost for Level III technology adequate for environmental protection is $500,000 or 0.68% of estimated national sales.

TABLE 88. COST SUMMARY FOR PRIMARY TUNGSTEN REFINING

Annual Production: Model Plant 300 MT Industry: 5,645MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.482
Digestion Residue	0.05

Cum. Unit Waste Disposal Costs:

Waste (Type)	Level I $/MT of Waste		Level I $/MT of Prod.	Level II $/MT of Waste		Level II $/MT of Prod.	Level III $/MT of Waste		Level III $/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$116.84	$46.74	$56.32	$116.84	$46.74	$56.32	$227.80	$91.12	$109.80
Annual Cost	34.59	13.84	16.67	60.50	24.20	72.99	77.82	31.13	86.17
Digestion Residue									
Capital Cost	1/		1/	73.33		3.67	73.33		3.67
Annual Cost	1/		1/	16.00		0.80	16.00		0.80
Total Capital Cost	-		56.32	-		59.99	-		113.47
Total Annual Cost	-		16.67	-		73.79	-		86.97

Cum. Industry Waste Disposal Costs ($ Million)

Waste (Type)	Level I Cap.	Level I Ann.	Level II Cap.	Level II Ann.	Level III Cap.	Level III Ann.
Sludge	$0.32	$0.09	$0.32	$0.41	$0.62	$0.49
Digestion Residue	1/	1/	0.02	0.01	0.02	0.01
Total	0.32	0.09	0.34	0.42	0.64	0.50

1973 Metal Price: $12,848/MT

Treatment Cost as a Percent of Metal Selling Price

Waste (Type)	Level I Cap.	Level I Ann.	Level II Cap.	Level II Ann.	Level III Cap.	Level III Ann.
Sludge	0.44%	0.13%	0.44%	0.56%	0.85%	0.67%
Digestion Residue	1/	1/	0.03	0.01	0.03	0.01
Total	0.44	0.13	0.47	0.57	0.88	0.68

1/ = negligible

9.0 PRIMARY SMELTING AND REFINING OF TIN (SIC 3339)

9.1 INDUSTRY CHARACTERIZATION

The only primary tin smelter in the United States is the Gulf Chemical & Metallurgical Company, Texas City, Texas plant. 1971, 1972 and 1973 productions as given by the U.S. Bureau of Mines (Ref. 2) were 4,000, 4,300 and 4,500 MT respectively. Marketable by-products of the plant are cathode melting dross, lead-tin alloy solder, copper-silver cement and ferric chloride solution. The average selling price of primary tin in 1973 was \$2.28/lb. Thus, national sales are estimated as \$22,572,000 in 1973.

9.2 WASTE CHARACTERIZATION

This section relates production processes and air and water effluent controls to the emanation of solid waste residuals. The disposition of these solid waste residuals is described.

9.2.1 Process Description Figure 13 is a flow diagram of the tin and associated by-product production at Texas City. Dispostion of process and emissions control solids is shown.

The major production process sequence employed in the production of tin ingot as shown in Figure 13 include the following operations:

1. Roasting of concentrate in a multiple hearth furnace to drive of SO_2 and produce an oxidized concentrate.
2. Acid leaching of roasted concentrate to remove iron, copper, silver and other metallic impurities.
3. Two stage smelting to produce 95% pure tin bullion and slag by-product.
4. Electrolytic refining of 95% tin bullion to produce 99.9% pure tin.

Effluent gases produced during roasting are treated by dry cyclone and electrostatic precipitator and sent to the acid leach. The degree of stack gas treatment for SO_2 removal is not known but may involve wet scrubbers with resultant scrubwater solids.

Acid leach is normally done with hydrochloric acid to remove iron, copper, silver and other metallic impurities. Leach solution is treated by thickening (with solids returned to process stream) and cementation to produce a copper/silver cement product and ferric chloride solution both of which are salable commodities.

In the two stage smelting operation, coal is used as a reductant. Dusts are removed from flue gases and returned to the smelting circuit. Iron

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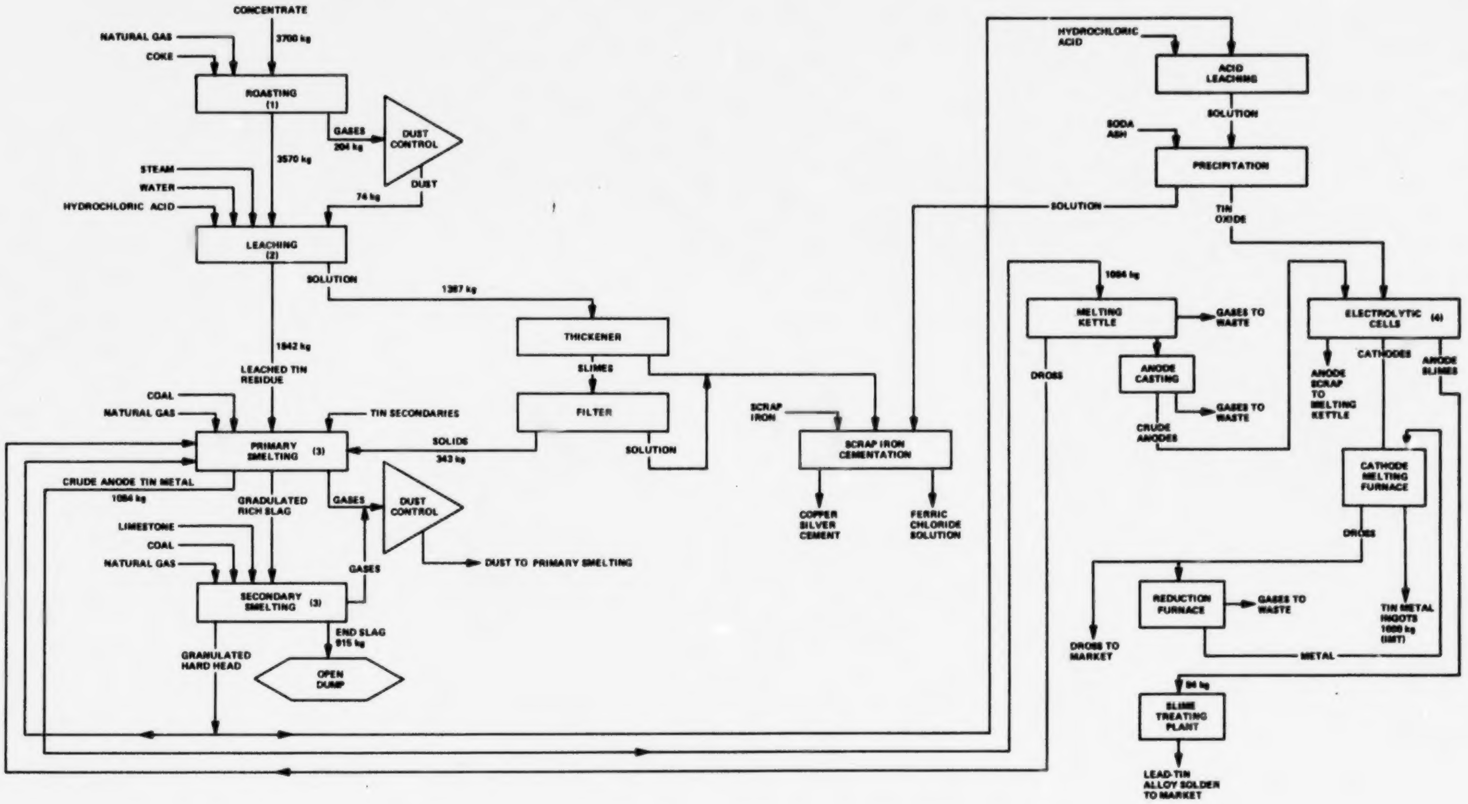

CONCENTRATE
3700 kg
NATURAL GAS
COKE
ROASTING (1)
GASES
204 kg
DUST CONTROL
DUST
74 kg
3570 kg
STEAM
WATER
HYDROCHLORIC ACID
LEACHING (2)
SOLUTION
1387 kg
THICKENER
SLIMES
FILTER
SOLUTION
1842 kg
LEACHED TIN RESIDUE
COAL
NATURAL GAS
TIN SECONDARIES
PRIMARY SMELTING (3)
SOLIDS
343 kg
CRUDE ANODE TIN METAL
1084 kg
GRADULATED RICH SLAG
GASES
DUST CONTROL
DUST TO PRIMARY SMELTING
LIMESTONE
COAL
NATURAL GAS
GASES
SECONDARY SMELTING (3)
END SLAG
915 kg
GRANULATED HARD HEAD
OPEN DUMP
SCRAP IRON
SCRAP IRON CEMENTATION
COPPER SILVER CEMENT
FERRIC CHLORIDE SOLUTION
HYDROCHLORIC ACID
ACID LEACHING
SOLUTION
SODA ASH
PRECIPITATION
SOLUTION
TIN OXIDE
1084 kg
MELTING KETTLE
GASES TO WASTE
DROSS
ANODE CASTING
GASES TO WASTE
CRUDE ANODES
ELECTROLYTIC CELLS (4)
ANODE SCRAP TO MELTING KETTLE
CATHODES
ANODE SLIMES
CATHODE MELTING FURNACE
DROSS
REDUCTION FURNACE
GASES TO WASTE
DROSS TO MARKET
TIN METAL INGOTS 1000 kg (1MT)
METAL
96 kg
SLIME TREATING PLANT
LEAD-TIN ALLOY SOLDER TO MARKET

Figure 13 PRIMARY TIN SMELTING AND REFINING

tin alloys are treated, in part, by acid leach to provide tin oxide and additional ferric chloride solution for sale. Tin metal is routed to electrolytic cells for further purification.

Electrolytic refining of crude tin from the smelter is carried out in sulfuric acid solution to produce pure tin cathodes. The cathodes are melted and cast into ingots. Drosses from the melting operation are either sold for custom recovery or returned to the process stream. Anode slimes from the electrolytic circuit are treated separately to produce a lead/tin alloy; no details on wastes from slime treatment is currently available. The electrolytic circuit is reported to be operable without the necessity for periodic bleed of electrolyte solution.

9.2.2 Description of Waste Streams

Smelting Slag From the previous discussions of tin production it is seen that much of the residue from tin ore concentrate leaching and electrolytic refining are further processed for recovery of copper, silver, lead and other by-products. Residuals of no value and a small quantity of nonrecoverable metallics end up in a discard slag which is open dumped. From long term materials balance studies at the Texas City smelter (Ref. 5) it has been shown that 26% of smelter input or 915 Kg/MT of electrolytic tin output ends up as discard slag.

Discard slag is in granulated form and therefore individual aggregates are expected to range in size from sand size grains to a few inches in diameter. The tin content of discard slag is said to average 1% (Ref. 5). Lead content is given as .01% (100 ppm). Iron content is given as 10.3%. Traces of antimony are reported. Calspan was not able to obtain slag samples for further analyses. However, mineralology of input concentrates indicate that copper, arsenic, and zinc should be present in trace amounts. In the absence of solubility data discard slag is considered potentially hazardous because of lead and other heavy metal content and fine particle size resulting from granulation.

Table 89 gives the generation factor for discard slag from the tin smelter along with estimated concentration of tin and lead. Arsenic, copper, zinc and antimony are believed present also but since slag samples were not made available their concentrations could not be ascertained. Table 90 gives the quantities of slag and hazardous constituents thereof for an annual production of 4,000 MT tin metal.

Dusts. Dusts collected in emissions control devices are recycled to the process.

9.2.3 Waste Quantities Based on production of 4,000 MT/year in recent years and using a factor of 915 Kg discard slag/MT, it is estimated that 3,660 MT of slag was land disposed from the Texas City smelter in 1974. This slag will contain 0.36 MT of lead and 36.6 MT of tin, potentially hazardous substances. Production and consequent slag generation is expected to remain at present levels through 1985 since production has varied little from 1970 to 1973, (Ref. 2). Table 91 gives estimated slag disposal and toxic constituents

TABLE 89 WASTE RESIDUAL FACTORS - PRIMARY TIN SMELTING AND REFINING

Type of Waste	Generation Factor kg/MT	Concentration Factor (ppm)				
		Sn	Pb	Sb	As	Zn
Discard slag	915	10,000	100	NK	NK	NK

NK = presence suspected but concentration unknown.

TABLE 90 YEARLY GENERATION OF WASTE RESIDUALS, FROM TYPICAL PLANT, PRIMARY TIN SMELTING AND REFINING

Type of Waste	Total Waste Quantity kg/MT	Quantity of Potentially Hazardous Constituents (MT)						
		As	Pb	Cu	Zn	Sn	Sb	etc.
Discard slag	3660	NK	0.36	NK	NK	36.6	NK	

NK = not known

* Quantities calculated from generation and concentration factors given in Table 89 based on annual production of 4,000 MT. Divide by 350 to obtain daily quantities, multiply by 1.1 to convert to short tons.

Table 91

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM PRIMARY TIN SMELTING AND REFINING, 1974, 1977, 1983 (DISCARD SLAG) METRIC TONS *

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS					
					As	Cu	Pb	Sb	Sn	Zn
TEXAS	3,660	3,660	> 37	OPEN DUMP	NK	NK	0.36	NK	36.6	NK
EPA REGION VI	3,660	3,660	> 37	OPEN DUMP	NK	NK	0.36	NK	36.6	NK
NATIONAL					NK	NK	0.36	NK	36.6	NK

NK – NOT KNOWN

* MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORP.

thereof for 1974, 1977, and 1983.

9.3 TREATMENT AND DISPOSAL TECHNOLOGY

9.3.1 Current Waste Treatment and Disposal Technology

The current method of waste disposal is open dumping of discard slag. In the absence of further data to indicate otherwise, this waste is considered potentially hazardous. Levels of technology for treatment and disposal of slag are described in the following sections.

9.3.2 Present treatment and Disposal Technology (Level I)

Discard Slag In the absence of information from the plant it is assumed that the present method of slag disposal is open dumping. In the absence of samples for solubility tests it must be assumed that lead or other toxic metals may be released in significant concentrations. If solubilized heavy metals percolate through permeable soils to ground water protective measures as described under Level III will be needed.

Dusts Recycling of dusts to the process is done and is a sound environmental practice.

9.3.3 Best Technology Currently Employed (Level II)

Level II practices are the same as Level I for slag and dust.

9.3.4 Technology To Provide Adequate Health and Environmental Protection (Level III)

Discard Slag If leaching of solubilized toxic metals to ground water or runoff occurs then protective measures are needed. These measures include provision for collection and treatment of slag pile runoff and sealing of the ground surface at slag disposal sites to prevent percolation of leachate.

Dusts Recycling of dusts to the process is an environmentally sound and commendable practice.

Features of Levels I, II and III treatment and disposal technologies including assessment of the hazardous nature of residuals and the methods and adequacy of treatment and disposal methods are summarized in Table 92.

9.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The cost of implementing this technology for a typical tin smelter is estimated here.

Table 92

Treatment and Disposal Technology Levels

Smelting and Refining Category PRIMARY TIN SIC 3339

DISCARD SLAG

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Granulated to sand size to ~ 3 inch diameter separates; oxides and silicates	Same as I	Same as I
Amount of Waste (kg/MT Product)	915 kg	915 kg	915 kg
Factors Affecting Hazardousness	Presence of lead, tin, copper, arsenic and other heavy metals	Same as I	Same as I
Treatment/ Disposal Technology	Open Dumping	Open Dumping	Sealing of ground at disposal area; collection of runoff
Estimate of # + % of Plants Using Technology	1 (100%)	1 (100%)	0

Table 92-(cont'd.)

SMELTING SLAG (continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if heavy metals are leached	Same as I	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	Possible groundwater and surface water contamination if heavy metals are leached	Same as I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	groundwater monitoring wells and runoff monitoring
Energy Requirements	Negligible	Negligible	Negligible

9.4.1 Cost of Present Treatment and Disposal Technology (Level I)

The plant produces 4,000 MT of tin annually which results in the generation of 3,660 MT of slag (915 kg/MT tin). The slag is disposed of on an on-site dump.

Loading and hauling are estimated to require 200 hrs/yr of front loader and truck time. An additional 20 hrs/yr of bulldozer time is assigned at the dump for spreading and compacting the slag.

The slag has an estimated density of 1675 Kg/m^3; thus 2,185 m^3 of waste are disposed annually. A .5 ha dump site is required to accommodate 20 years of slag assuming it is piled to an average height of 10 m. Capital and annual costs are summarized in Table 93.

9.4.2 Cost of Best Technology Currently Employed (Level II)

The technology and costs for Level II technology are the same as Level I.

9.4.3 Cost of Technology to Provide Adequate Health & Environmental Protection (Level III)

The land is sealed at the slag dump, collection ditches, and pump and piping are installed. The incurred and incremental costs are:

Capital cost

Land sealing	$ 10,000
Collection ditches	1,070
Pump and piping	8,210
Total	$ 19,280

Annual Cost

Construction Amortization	$ 1,285
Equipment Amortization	1,310
Construction Maintenance & Repair	330
Equipment Maintenance & Repair	410
Energy	
Electricity	30
Insurance	195
Total	$ 3,560

Costs of Levels I, II and III treatment and disposal technology are summarized in Table 94 for smelter discard slag. Costs are given in dollars per metric ton of waste and dollars per metric ton of tin product. Costs for each level of technology are also expressed as a percent of metal selling price.

TABLE 93

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY PRIMARY TIN SMELTING AND REFINING PLANT

CAPITAL COST

Equipment	Slag
Equipment	
Truck (10%)	$ 2,500
Frontloader (10%)	2,000
Bulldozer (1%)	160
Dump	
Survey	375
Land	875
Total	$ 5,850

ANNUAL COST

Land	$ 90
Amortization	
Construction	35
Equipment	740
Operating Personnel	4,565
Repair and Maintenance	
Construction	-----
Equipment	235
Energy	
Fuel	565
Electricity	55
Taxes	20
Insurance	60
Total	$ 6,365

TABLE 94 COST SUMMARY FOR PRIMARY TIN SMELTING AND REFINING

Annual Production: Model Plant 4,000 MT Industry 4,572 MT

Waste (Type)	Amount (MT;MT of Production)
Slag	0.92

Cumulative Unit Waste Disposal Costs:

	Level					
Waste (Type)	I		II		III	
	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.
Slag						
Capital Cost	$1.60	$1.46	$1.60	$1.46	$6.87	$6.28
Annual Cost	1.73	1.58	1.73	1.58	2.70	2.47
Total Capital Cost	-	1.46	-	1.46	-	6.28
Total Annual Cost	-	1.58	-	1.58	-	2.47

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
	I		II		III	
Waste (Type)	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	0.01	0.01	0.01	0.01	0.03	0.011
Total	0.01	0.01	0.01	0.01	0.03	0.011

1973 Metal Price: $5,006/MT

Treatment Cost As Percent of Metal Selling Price

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	0.03%	0.03%	0.03%	0.03%	0.13%	0.05%
Total	0.03	0.03	0.03	0.03	0.13	0.05

Total industry costs for Levels I, II and III treatment and disposal technologies have been estimated in 1973 dollars and are presented in Table 94 The annualized industry costs for Levels I and II treatment and disposal technology is estimated as 10,000 or approximately 0.03% of estimated 1973 material sales value. The annualized industry cost for Level III technology (i.e., adequate for environmental protection) is estimated as 11,000 or 0.05% of estimated 1973 national sales value.

10. PRIMARY SMELTING AND REFINING OF MAGNESIUM (SIC 3339)

Reported production of primary magnesium metal in the United States has steadily increased from 89,246 MT in 1968 to 109,611 MT in 1972 (Ref. 1). Nearly the entire U.S. production of magnesium in 1972 was attributed to the Dow Chemical Company at Freeport, Texas. American Magnesium Company, Snyder, Texas and NL Industries, Rowley, Utah also produced minor amounts of metal. Total domestic production capacity for magnesium metal is expected to be about 213,000 MT/year by 1975 (Ref. 1). The 1973 production of primary magnesium metal was 124,538 MT. The selling price was 38.25 cents/lb., (Ref.2). Thus, national sales can be estimated as $105,017,000. Because of insufficient data collection it was not possible to quantify the types of wastes associated with primary magnesium production or the costs associated with waste disposal. Consideration of the raw materials and processes used in magnesium metal production indicates the absence of hazardous components in significant concentrations or volumes. Very little or none of this industries' residuals will be land disposed.

Production processes and qualitative assessment of the nature and disposition of wastes are presented in the following paragraphs.

The major method employed for production of magnesium metal is electrolysis of seawater (Dow Chemical, Freeport, Texas) or other high salt water such as the Great Salt Lake (NL Industries, Rowley, Utah). This process is shown in Figure 14. Minor production of magnesium is from dolomite limestone by reduction with ferrosilicon at high temperatures (Northwest Alloys, Inc., Addy, Washington). This process is shown in Figure 15.

Neither of the above processes can be expected to produce significant quantities of potentially hazardous wastes. Seawater which is the raw material for the Dow process contains less than 0.1 ppm of each of a number of heavy metals and is discharged back into the ocean after precipitation of magnesium as magnesium hydroxide. The principal waste is a filter cake consisting mainly of sodium chloride and calcium sulfate which are nonhazardous. This cake is slurried off for waste disposal in the ocean.

The only significant solid wastes from the ferrosilicon process are slags from the retort and smelting furnaces. These slags are generated at rates of 7055 kg and 64 kg per metric ton of magnesium produced for retort and smelting furnaces, respectively. The slags are open dumped.

Dolomitic limestone which is the principal raw material for production of magnesium by the ferrosilicon process contains only small amounts of trace metals. Typical analysis of limestone for trace metals is given in Table 95.

Limestone is used extensively for soil pH adjustment in agriculture. Discard slag would not be expected to differ significantly in concentration of heavy metals and, therefore, is considered nonhazardous.

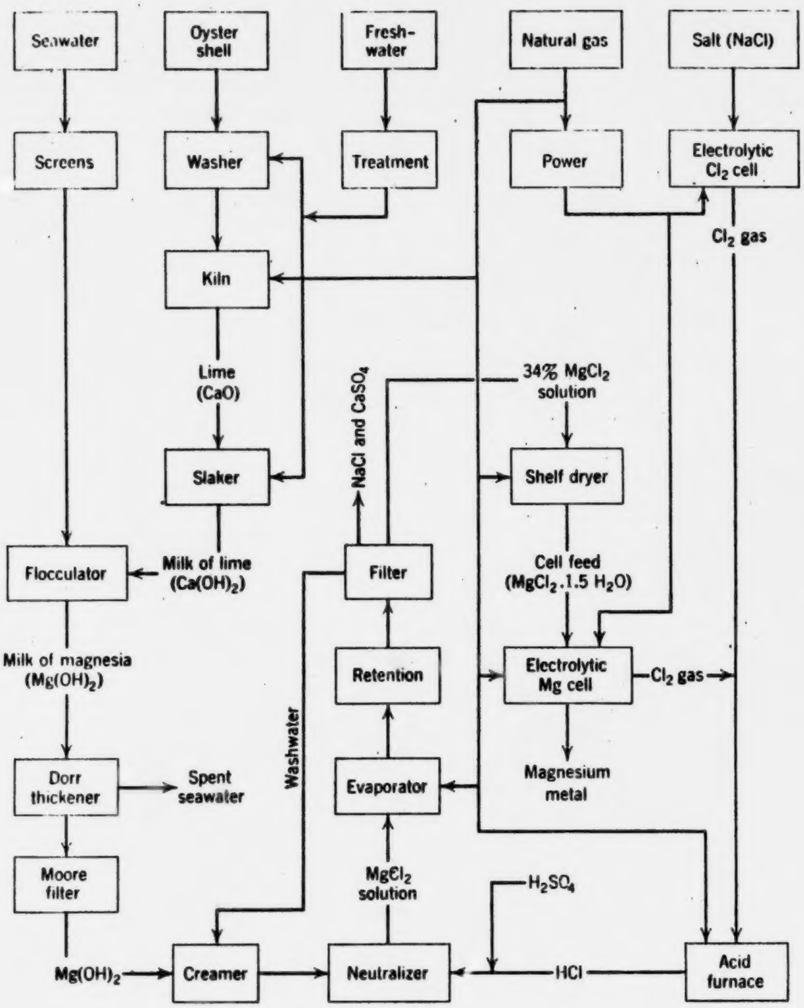

Seawater
Oyster shell
Fresh-water
Natural gas
Salt (NaCl)
Screens
Washer
Treatment
Power
Electrolytic Cl₂ cell
Cl₂ gas
Kiln
Lime (CaO)
NaCl and CaSO₄
34% MgCl₂ solution
Slaker
Shelf dryer
Flocculator
Milk of lime (Ca(OH)₂)
Filter
Cell feed (MgCl₂.1.5 H₂O)
Milk of magnesia (Mg(OH)₂)
Retention
Electrolytic Mg cell
Cl₂ gas
Washwater
Dorr thickener
Spent seawater
Evaporator
Magnesium metal
Moore filter
MgCl₂ solution
H₂SO₄
Mg(OH)₂
Creamer
Neutralizer
HCl
Acid furnace

FIGURE 14 PRODUCTION OF MAGNESIUM FROM SEAWATER

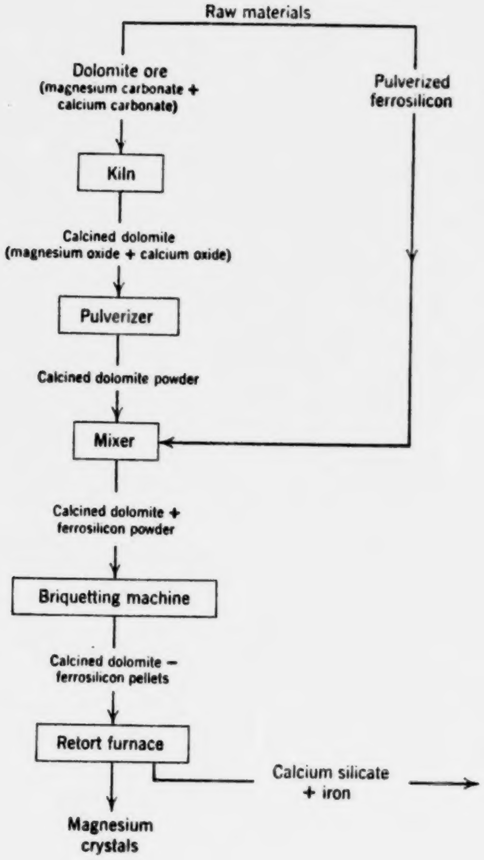

Raw materials
Dolomite ore
(magnesium carbonate + calcium carbonate)
Pulverized ferrosilicon
Kiln
Calcined dolomite
(magnesium oxide + calcium oxide)
Pulverizer
Calcined dolomite powder
Mixer
Calcined dolomite + ferrosilicon powder
Briquetting machine
Calcined dolomite – ferrosilicon pellets
Retort furnace
Calcium silicate + iron
Magnesium crystals

FIGURE 15 PRODUCTION OF MAGNESIUM BY FERROSILICON PROCESS

TABLE 95

TRACE ELEMENT COMPOSITION OF LIMESTONE

Element	Concentration (ppm)
As	1
Be	1
Cd	0.04
Cr	11
Cu	4
Pb	9
Hg	.04
Se	0.08
Zn	20

11.0 PRIMARY SMELTING AND REFINING OF CADMIUM (SIC 3339)

There is no discrete primary cadmium industry in the United States. Primary cadmium metal and compounds are byproduct materials produced by companies principally engaged in the recovery of zinc either as a major value or as a byproduct itself. This is because there is no separate ore of cadmium and because it occurs in significant quantities in nature exclusively in association with zinc sulfides. Major cadmium producers in the United States are shown in Table 96. Total cadmium metal production in the country is small. Production for the years 1971 to 1975 was as follows: (Ref. 10)

1971	3580 MT
1972	3765 MT
1973	3400 MT
1974	3020 MT
1975	1990 MT

TABLE 96 Producers of Cadmium Metal

American Smelting & Refining Company	Denver, Colorado
American Smelting & Refining Company	Corpus Christi, Texas
Amax Zinc Company	East St. Louis, Illinois
The Bunker Hill Company	Kellogg, Idaho
National Zinc Company	Bartlesville, Oklahoma
The New Jersey Zinc Company	Palmerton, Pennsylvania
St. Joe Minerals Corporation	Monaca, Pennsylvania
United Refining & Smelting Company	Franklin Park, Illinois

These figures include secondary cadmium as well as primary since individual breakdowns are not published. Table 97 gives the state and regional distribution of cadmium production for the year 1973. Plant capacities are not available.

Examination of the various alternative methods for recovery of cadmium metal from zinc smelters reveals that relatively small quantities of nonrecyclable waste are produced. These wastes are considered potentially hazardous, however. Since cadmium production is always a minor operation at primary zinc smelters and occassionally at primary lead or copper smelters the waste streams become minor inputs to the larger waste streams. These waste streams are covered previously in this report. Cadmium recovery wastes will always report either as sludges or dusts. As such, lined lagoons or sealed soil areas would be required for their disposal as stipulated in the sections on primary zinc, lead and copper. For these reasons and unavailability of

TABLE 97

GEOGRAPHICAL DISTRIBUTION OF PRIMARY CADMIUM PRODUCTION

State	No. of Plants	Plant Distribution by Production (MT)
Colorado	1	NA
Idaho	1	522 (1973)
Illinois	1	123 (1973)
Oklahoma	1	NA
Pennsylvania	2	662 (1973)
Texas	1	NA
EPA Region		
3	2	662 (1973)
5	1	123 (1973)
6	2	NA
8	1	NA
10	1	552 (1973)
National	7	3440 (1973)

data on quantification of cadmium wastes from the many alternative processes for recovery details on quantities of cadmium plant wastes and costs of treatment and disposal are not presented in this section.

Contained in the following is a description of the alternative methods for cadmium metal production and identification of possible sources of potentially hazardous residuals which may be disposed on land.

Process Description

The starting material for cadmium-recovery processes are by-products of other metal-production operations, all involving zinc. These are:

(1) Fumes and dusts from the roasting and sintering of zinc concentrates;

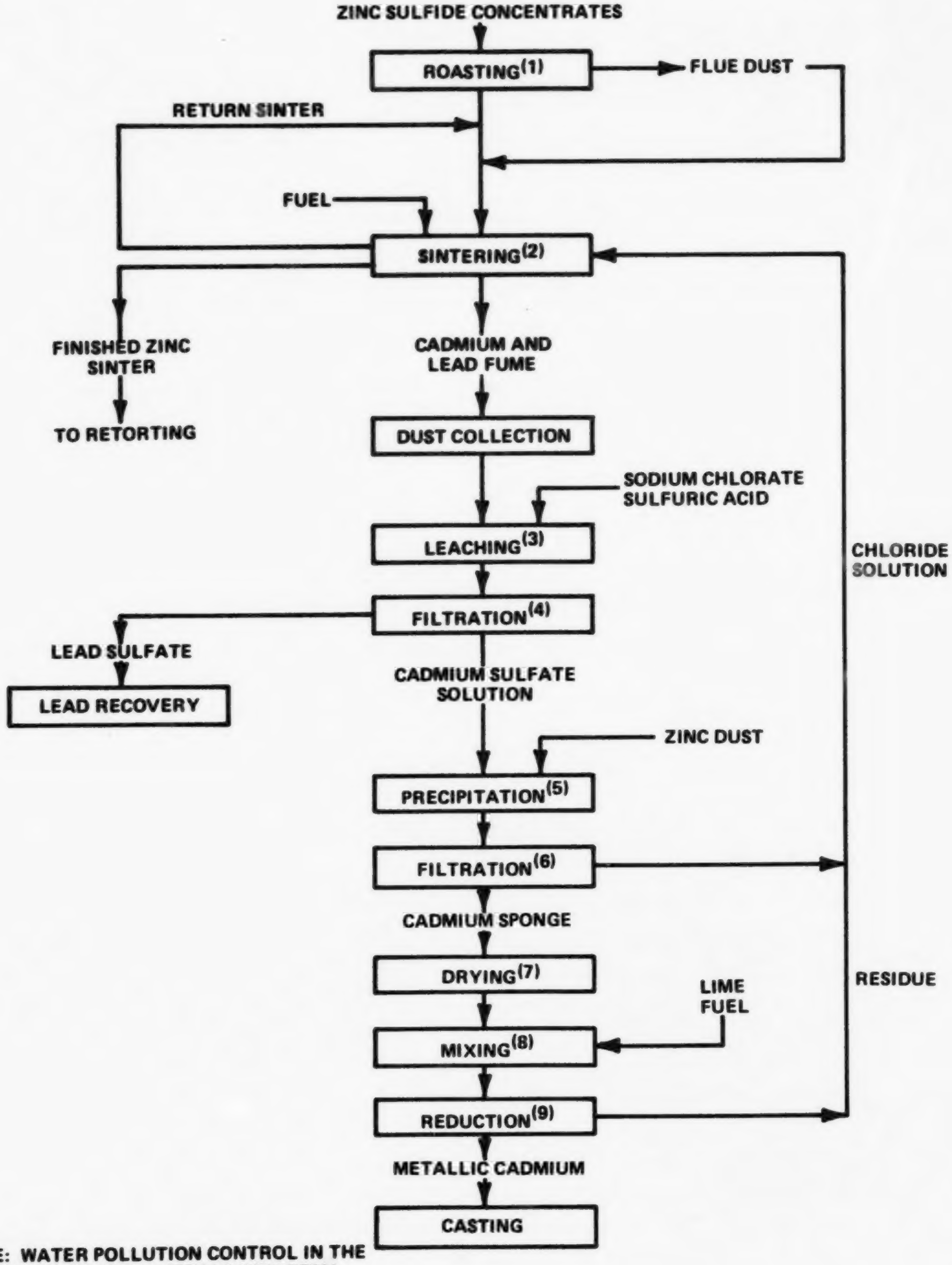
(2) Zinc metal containing relatively high concentrations of lead and cadmium;

(3) Dusts from the smelting of lead-zinc ores or copper lead-zinc ores;

(4) Purification sludge from zinc electrowinning plants.

Cadmium from Zinc Concentrates. In the recovery of zinc from zinc concentrates by the retort process, zinc sulfide concentrates are roasted, sintered (see zinc section), and introduced into retorts with reducing agents such as coal or coke. When the mixture is heated, the zinc oxide is reduced to zinc metal which is volatilized, condensed, and cast into ingots. Unless special provisions are taken, much of the cadmium and lead almost always present in zinc concentrates would accompany the zinc. Figure 16 outlines a process used to separate these elements from the zinc and at the same time recover cadmium.

Roasting is done in multiple-hearth furnaces under oxidizing conditions to eliminate most of the sulfur (1). At the temperature of the roasting operation, cadmium tends to volatilize preferentially and concentrates to some extent in the flue dust. This is collected in suitable equipment such as cyclones, baghouses, and electrostatic precipitators and is returned to join the roasted product. The roasted concentrates and flue dust are then charged onto the surface of a sintering machine (2) which agglomerates the calcine, removes residual sulfur, and removes most of the lead and cadmium from the zinc. Lead-cadmium removal is accomplished by chloridizing the charge with the zinc chloride solution returned from filtration step or by the addition of sodium chloride solution. Under proper conditions of temperature and chloride addition, cadmium and lead chloride volatilize from the charge and are drawn down through the bed and into the fume-collecting facilities for subsequent treatment. The sintered zinc oxide product is sent to the retorting furnaces for the recovery of zinc metal.


ZINC SULFIDE CONCENTRATES
ROASTING(1)
FLUE DUST
RETURN SINTER
FUEL
SINTERING(2)
FINISHED ZINC SINTER
TO RETORTING
CADMIUM AND LEAD FUME
DUST COLLECTION
SODIUM CHLORATE
SULFURIC ACID
LEACHING(3)
CHLORIDE SOLUTION
FILTRATION(4)
LEAD SULFATE
LEAD RECOVERY
CADMIUM SULFATE SOLUTION
ZINC DUST
PRECIPITATION(5)
FILTRATION(6)
CADMIUM SPONGE
DRYING(7)
LIME
FUEL
RESIDUE
MIXING(8)
REDUCTION(9)
METALLIC CADMIUM
CASTING

SOURCE: WATER POLLUTION CONTROL IN THE PRIMARY NONFERROUS METALS INDUSTRY - V/I COPPER, ZINC AND LEAD INDUSTRIES, EPA-R2-73-247a, SEPTEMBER 1973

Figure 16 CADMIUM RECOVERY IN THE ZINC RETORT PROCESS

After collection, the cadmium and lead fumes and dusts are leached with dilute sulfuric acid and sodium chlorate (3) to ensure the complete dissolution of cadmium sulfide which would otherwise remain undissolved. In the leaching operation, cadmium and lead are converted to sulfates and/or chlorides. Cadmium sulfate and chloride remain in solution, but lead is almost completely converted to insoluble lead sulfate. This compound, together with other insoluble materials (quartz, silicates, etc.), is filtered out and sent to lead recovery, (4). The solution of cadmium sulfate (and/or chloride) is then treated with powdered zinc to precipitate cadmium (5). The slurry is filtered (6). The solution from filtration, containing practically all of the zinc added and about 10 percent of the cadmium as chlorides and sulfates, is returned to the sintering operation.

The cadmium filter cake, contains about 30 percent moisture and only small amounts of lead and zinc. This sponge is then steam dried, (7) and mixed with coal or coke and lime, (8) and transferred to a retort furnace where the cadmium is reduced to metal, vaporized, recondensed, and cast into required shapes. (9) Residue from the reduction furnace or retort containing some cadmium is returned to the sintering operation.

Cadmium from Zinc Metal. Some zinc made by the retort process contains both cadmium and lead in appreciable quantities. An example is the so-called prime Western zinc which may contain about 1.5 percent lead and several tenths of a percent cadmium. Such a material can be highly refined by the process shown in Figure 17. This process involves two fractionation stages, each at different temperatures. In the first, (1) operated at a temperature in excess of 1000 to 1150 C, cadmium and zinc are boiled off, leaving a high lead-zinc alloy. The cadmium-zinc vapors are condensed and refractionated at about 800 C. (2) In this operation, cadmium is volatilized and a high-purity zinc product is withdrawn at the bottom of the fractionation column.

The cadmium product from this process generally requires further treatment by additional volatilization or chemical means.

Cadmium Recovery From Lead Smelter Dusts. Smelter dusts, which customarily contain about 5 percent cadmium or less, as produced, may be treated in several ways to recover cadmium. Figure 18 shows three typical routes. In the first route (left side of Figure 18), the starting material is dusts from the smelter baghouses. In the case of burned baghouse dusts, (1) the dusts are collected in bag filters and dropped into masonry or refractory chambers called cellars where they are ignited and allowed to oxidize by burning or smoldering. (2) This material, after continued burning, will contain about 15 percent cadmium, 15 percent zinc, and 50 percent or more of lead. This is sent to a lead smelter where it is refumed (3) in a so-called "Godfrey" furnace with fuel and limestone at a temperature regulated to more or less selectively volatilize cadmium. The fume collected in a baghouse typically may contain up to 55 percent cadmium and about 15 percent each of lead and zinc. This material is generally sent to an electrolytic zinc refinery (4) for further treatment.

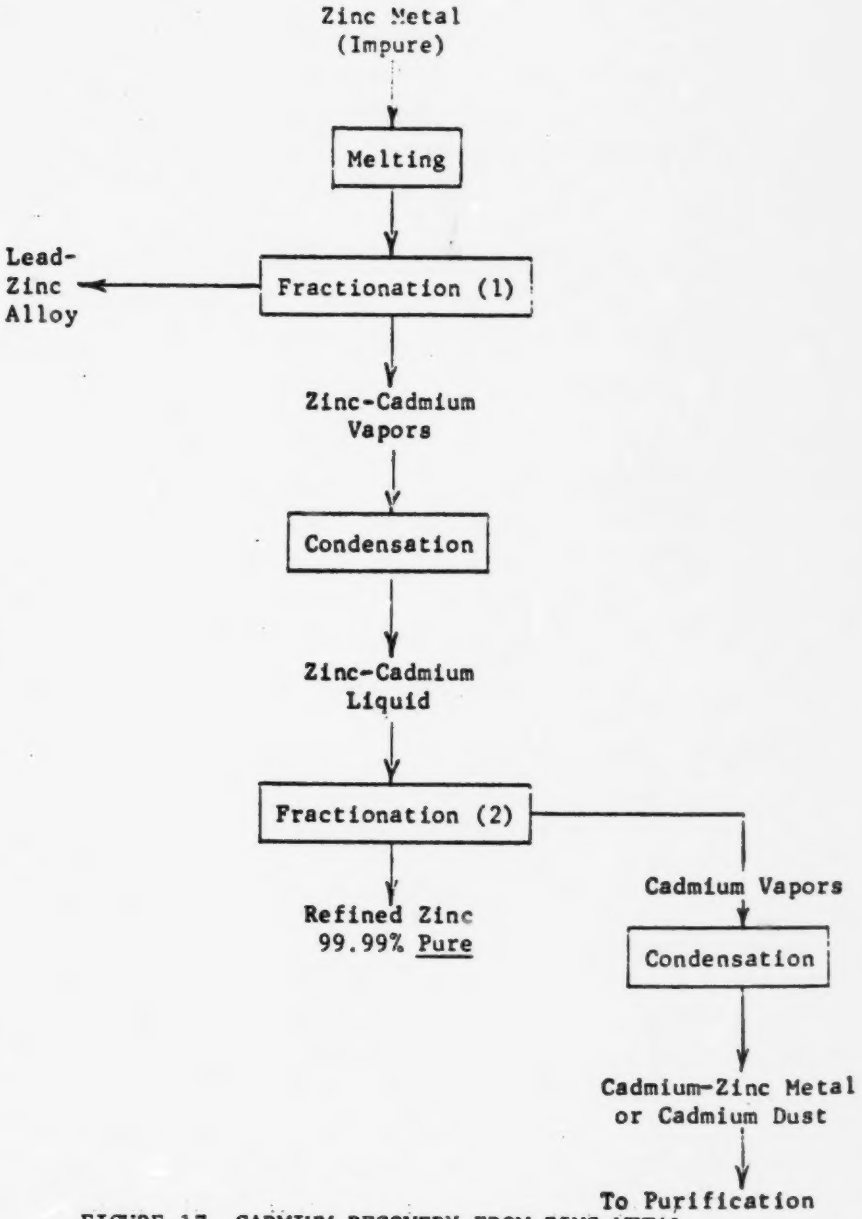
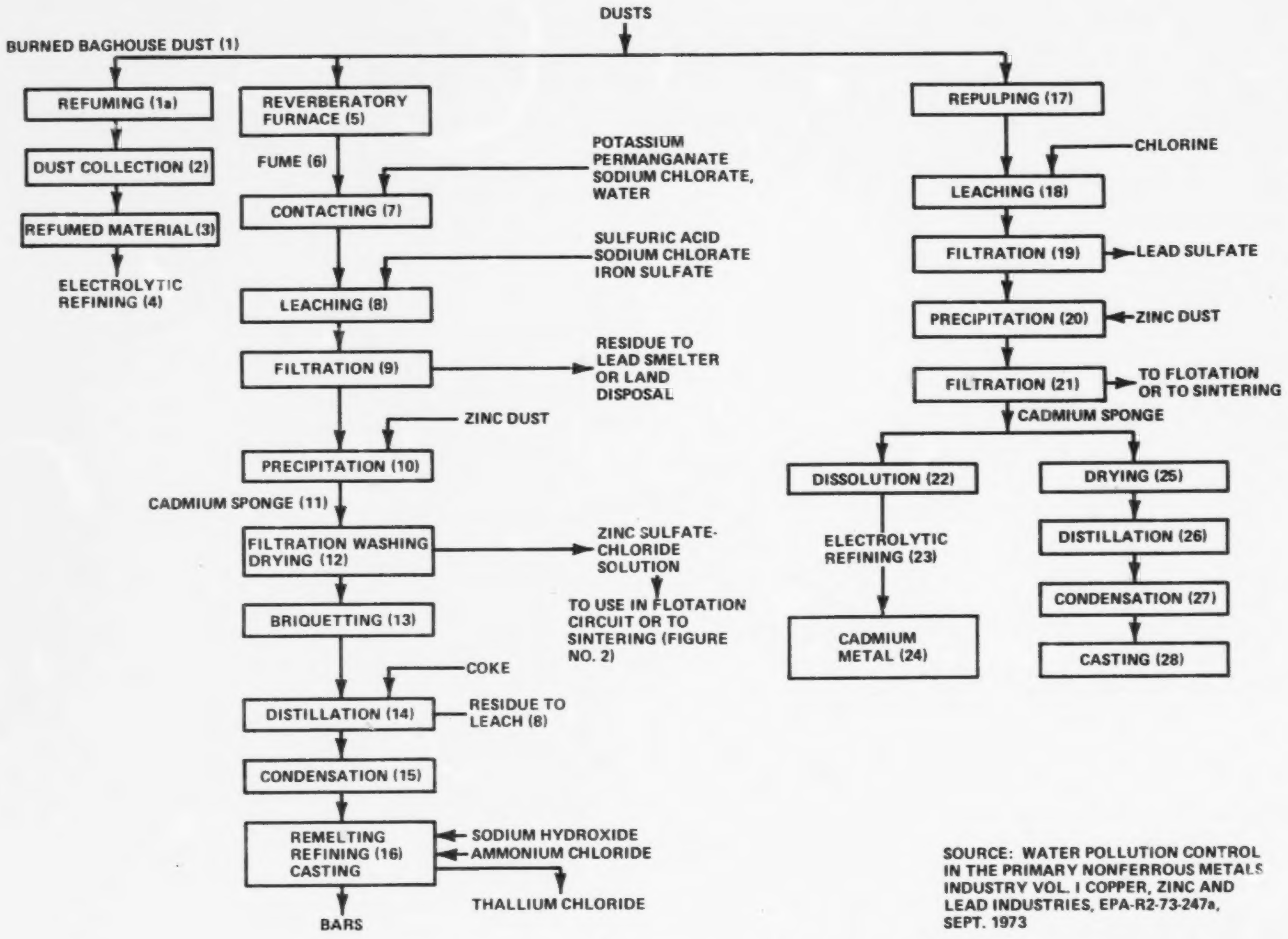

Zinc Metal (Impure)
Melting
Lead-Zinc Alloy
Fractionation (1)
Zinc-Cadmium Vapors
Condensation
Zinc-Cadmium Liquid
Fractionation (2)
Refined Zinc 99.99% Pure
Cadmium Vapors
Condensation
Cadmium-Zinc Metal or Cadmium Dust
To Purification

FIGURE 17 CADMIUM RECOVERY FROM ZINC METAL

SOURCE: Water Pollution Control in The Primary Nonferrous Metals Industry, VOL.I Copper, Zinc and Lead Industries, EPA-R2-73-247a, Sept. 1973


DUSTS
BURNED BAGHOUSE DUST (1)
REFUMING (1a)
DUST COLLECTION (2)
REFUMED MATERIAL (3)
ELECTROLYTIC REFINING (4)
REVERBERATORY FURNACE (5)
FUME (6)
CONTACTING (7)
POTASSIUM PERMANGANATE SODIUM CHLORATE, WATER
LEACHING (8)
SULFURIC ACID SODIUM CHLORATE IRON SULFATE
FILTRATION (9)
RESIDUE TO LEAD SMELTER OR LAND DISPOSAL
ZINC DUST
PRECIPITATION (10)
CADMIUM SPONGE (11)
FILTRATION WASHING DRYING (12)
ZINC SULFATE-CHLORIDE SOLUTION
TO USE IN FLOTATION CIRCUIT OR TO SINTERING (FIGURE NO. 2)
BRIQUETTING (13)
COKE
DISTILLATION (14)
RESIDUE TO LEACH (8)
CONDENSATION (15)
REMELTING REFINING (16) CASTING
SODIUM HYDROXIDE
AMMONIUM CHLORIDE
THALLIUM CHLORIDE
BARS
REPULPING (17)
CHLORINE
LEACHING (18)
FILTRATION (19)
LEAD SULFATE
PRECIPITATION (20)
ZINC DUST
FILTRATION (21)
TO FLOTATION OR TO SINTERING
CADMIUM SPONGE
DISSOLUTION (22)
ELECTROLYTIC REFINING (23)
CADMIUM METAL (24)
DRYING (25)
DISTILLATION (26)
CONDENSATION (27)
CASTING (28)

SOURCE: WATER POLLUTION CONTROL IN THE PRIMARY NONFERROUS METALS INDUSTRY VOL. I COPPER, ZINC AND LEAD INDUSTRIES, EPA-R2-73-247a, SEPT. 1973

Figure 18 CADMIUM RECOVERY FROM LEAD, ZINC, OR COPPER SMELTER DUSTS

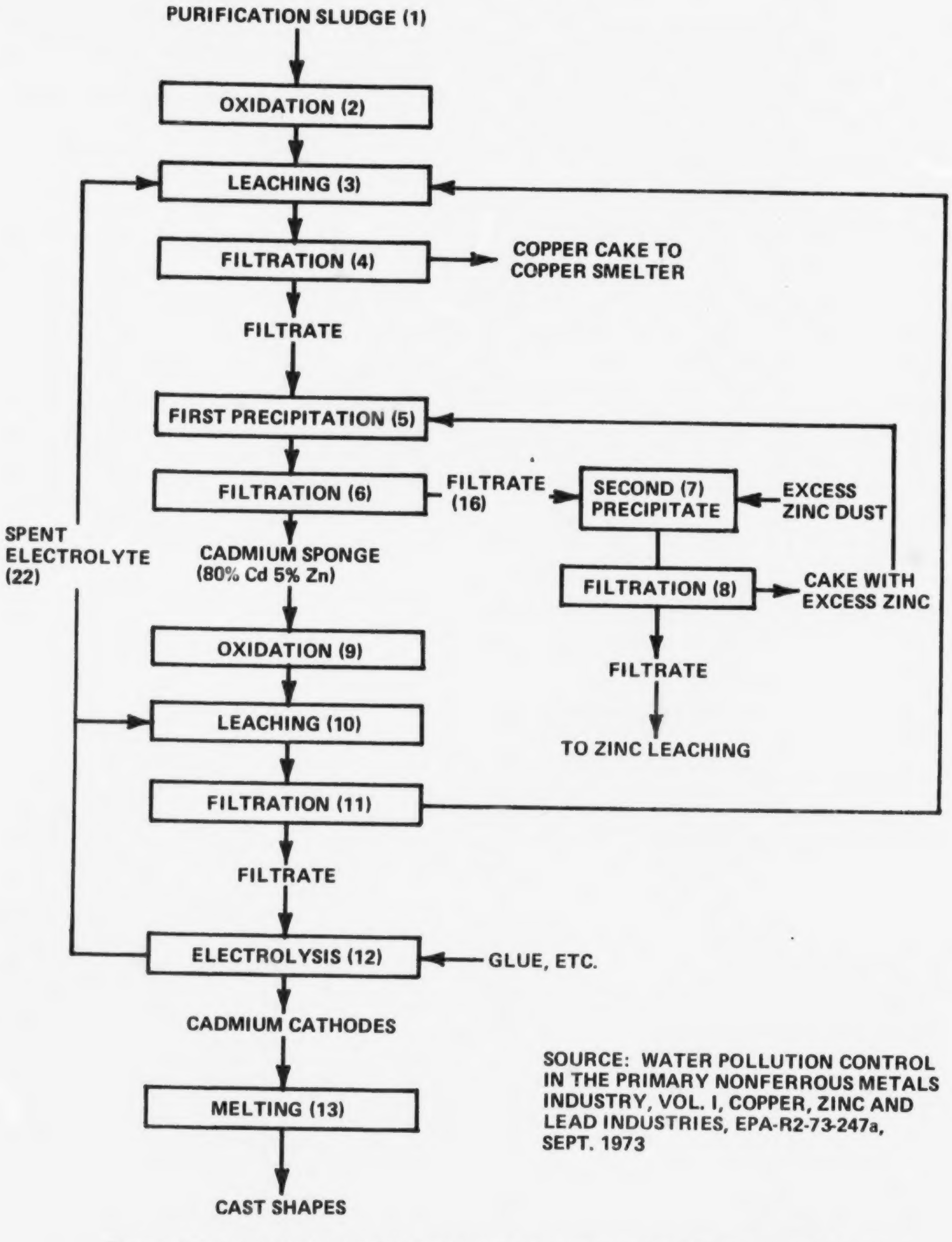
Another route is shown in the center column of Figure 18. By this route, the baghouse dust containing about 3 to 4 percent cadmium is refumed in a reverberatory furnace (5) in several batch stages in order to obtain a fumed product of optimum grade for subsequent processing. This fume, (6) which may contain 40 to 45 percent cadmium is contacted with water and oxidizing agents $KMnO_4$ and $NaClO_3$ (7) in a ball mill and leached by agitation with additional sodium chlorate, ferrous sulfate, and sulfuric acid. (8) The purpose of the oxidizing agents and the ferrous sulfate is to ensure that cadmium is all dissolved and that arsenic is oxidized and precipitated as ferric arsenate. The slurry is filtered; (9) the residue, consisting of lead sulfate and ferric arsenate, is sent back to lead smelters.

Precipitation with zinc dust under controlled conditions of acidity (10) produces a fairly high-grade "sponge". (11) This is filtered, washed by repulping with water, refiltered, and dried. (12) The filter cake is then granulated and formed into small briquettes. (13) The briquettes are then retorted batchwise in graphite crucibles to volatilize cadmium. (14) The cadmium vapors are condensed (15) and the liquid metal cast into bars. These are remelted under a layer of sodium hydroxide flux is removed and ammonium chloride is added to remove thallium from the cadmium. The cadmium is then recovered with sodium hydroxide flux and the purified metal, which may contain 99.95 percent of cadmium, is cast into bars.

The third route, shown at the right side of Figure 18 is also applicable to lead and zinc smelter fumes. In this process the fume or dust is leached by repulping in water (17) and the slurry is treated with chlorine gas. (18) Subsequent steps of filtration, (19) precipitation, (20) and isolation of cadmium sponge are similar to those previously described. The resultant sponge may be subsequently treated by electrolytic or pyrometallurgical methods to produce pure cadmium.

Cadmium Recovery from Purification Sludge in Zinc Electrowinning. The electrowinning process for zinc is described in the section of this report dealing with that metal. The successful electrodeposition of zinc requires extremely pure solutions. Cadmium, among other metals, such as copper, nickel, cobalt, and iron, must be virtually completely removed. In zinc metallurgy, both copper and cadmium are eliminated from solution by treating the zinc electrolyte with powdered zinc. Generally this is done stagewise. In the first stage, only enough zinc metal is added to precipitate copper, which, reacts preferentially with the zinc powder. After removal of the copper precipitate by filtration, the zinc electrolyte is then treated with additional zinc dust to precipitate the cadmium. This precipitate, high in zinc, contains virtually all the cadmium originally in the electrolyte and constitutes the purification sludge. Its processing is shown in Figure 19 .

Purification sludge (1) is first allowed to oxidize in air (2) with or without heating, to render the cadmium readily soluble. The oxidized material is then leached with spent electrolyte from the electrolysis step. (3) After filtration (4) to remove insoluble copper introduced from the zinc electrolyte, the filtrate is stage-treated with zinc dust (5-8) to minimize


PURIFICATION SLUDGE (1)
OXIDATION (2)
LEACHING (3)
FILTRATION (4)
COPPER CAKE TO COPPER SMELTER
FILTRATE
FIRST PRECIPITATION (5)
FILTRATION (6)
FILTRATE (16)
SECOND (7) PRECIPITATE
EXCESS ZINC DUST
SPENT ELECTROLYTE (22)
CADMIUM SPONGE (80% Cd 5% Zn)
FILTRATION (8)
CAKE WITH EXCESS ZINC
OXIDATION (9)
FILTRATE
LEACHING (10)
TO ZINC LEACHING
FILTRATION (11)
FILTRATE
ELECTROLYSIS (12)
GLUE, ETC.
CADMIUM CATHODES
MELTING (13)
CAST SHAPES

SOURCE: WATER POLLUTION CONTROL IN THE PRIMARY NONFERROUS METALS INDUSTRY, VOL. I, COPPER, ZINC AND LEAD INDUSTRIES, EPA-R2-73-247a, SEPT. 1973

Figure 19 CADMIUM RECOVERY FROM PURIFICATION SLUDGE OF ZINC ELECTROWINNING OPERATIONS

zinc concentration in the cadmium sponge. This sponge containing about 80 percent cadmium with 5 percent zinc, is again oxidized by steam drying (9) to enhance the solubility of cadmium. It is leached in spent electrolyte (10) from the electrolysis circuit and filtered. (11) The filtrate, containing about 200 grams per liter of cadmium as sulfate, is then electrolyzed (12) to produce cadmium cathodes.

Electrolysis is carried out in banks of cells similar to zinc cells. After passage through the cells, the spent electrolyte is returned to various leaching stages in the circuit. Stripped cathode metal is washed, dried, and melted (13) under a flux and cast into various shapes.

It is seen from the flow diagrams and previous discussion that most residuals from cadmium production are recycled for metallic recovery. A sample of a cadmium plant filter cake residue was obtained from one zinc smelting facility (cadmium is recovered from flue dust). This waste amounted to 514 kg/day. Analyses of this land dumped waste is given in Table 98.

TABLE 98

ANALYSIS OF CADMIUM PLANT RESIDUE

Cd	280 ppm
Cr	24 ppm
Cu	1,150 ppm
Pb	215,000 ppm
Zn	39,000 ppm

In solubility tests described in Appendix B this waste was found to leach significant concentrations of lead (9.0 ppm) and zinc (4,000 ppm) and is considered a potentially hazardous waste as are all cadmium plant residues.

Since cadmium is recovered from the residual dusts or sludges primarily from zinc smelters and to a lesser extent at lead or copper smelters, and environmentally acceptable methods of residual disposal have been identified for these smelting categories the safe disposition of cadmium processing residuals is covered under these categories.

Any standards regarding the environmentally sound methods of residual disposal of cadmium plant wastes on land should be handled as part of overall standards for primary zinc smelters or specialty smelting plants.

12. PRIMARY SMELTING AND REFINING OF ARSENIC (SIC 3339)

The production of arsenic in the United States is almost exclusively as a by-product from the smelting of copper and lead ores. In the roasting and smelting of such ores, arsenic oxidized to As_2O_3, is volatilized, and is recovered in fumes and flue dust. These contain varying amounts of the trioxide (up to 30 percent) associated with the oxides of other metals such as copper, lead, antimony, zinc, or tin, and various other particulate materials.

The most marketable form of arsenic is the pure trioxide, with little demand for pure elemental metallic arsenic. Only about 3% of arsenic is produced as the metal. (Ref.2) In the United States, the only primary metal producer now making either arsenic trioxide or arsenic metal is the Tacoma smelter of the American Smelting and Refining Company. It recovers arsenic from the flue dust of its own and other copper and lead smelters. (Ref. 7) In earlier years Anaconda also produced arsenic in Montana. No production data have been published in recent years. The U.S. Bureau of Mines estimated U.S. production of arsenic compounds and metal as 9253 MT in 1973. Only 277 MT (i.e3%) is arsenic metal. The price of arsenic metal in 1973 was 1.00/lb, (Ref. 2). Thus, the national sales of arsenic for 1973 can be estimated as $554,000.

In order to characterize and quantify the total and hazardous waste associated with arsenic production it will be necessary to visit and sample such wastes (if there are any) from the one U.S. producer (ASARCO, Tacoma, Washington). At that time treatment and disposal technology and environmental adequacy thereof may be identified and associated costs estimated. A brief description of flue dust processing for arsenic recovery and associated wastes follows.

Processing consists of roasting the flue dusts to volatilize the contained As_2O_3, which concentrates in the fume from the roasting operation. The collected fume is again heated in a reverberatory furnace, and the fume from this operation is passed through a series of chambers to collect fractions of various purity from the volatilized furnace output.

Potentially hazardous arsenic compounds are always present in the collected dust emissions from copper, lead, zinc and other base metal roasting operations. These dusts are frequently recycled for recovery of arsenic or other metallics. Where they are not recycled, but disposed of or stored on land for significant time periods, attention must be given to possible leaching of arsenic. Sealing of disposal or storage areas may be necessary.

13.0 PRIMARY SMELTING AND REFINING OF SELENIUM AND TELLURIUM (SIC 3339)

13.1 INDUSTRY CHARACTERIZATION

The only, raw material for the recovery and refining of selenium and tellurium in the United States is electrolytic slimes, a by-product of copper refining. The precious metals gold, silver and platinum are also recovered from slimes.

Production of these metals is small. Production of selenium has ranged from 287 MT to 566 MT from 1968 to 1972, while production of tellurium has ranged from 55 MT to 117 MT over the same time period (Ref. 1).

U.S. producers of selenium and tellurium metal and compounds are listed in Table 99.

TABLE 99

United States Producers of Selenium and Tellurium Metals and Compounds

	Se	Te
American Metal Climax, Carteret, N.J.	X	X
American Smelting and Refining Co., Baltimore, Md.	X	X
International Smelting and Refining Co., Perth Amboy, N.J.		X
Kennecott Copper Corp., Garfield, Utah	X	
Anaconda Co., Perth Amboy, N.J.	X	X

Major electrolytic copper refiners ship anode muds and slimes to the above locations for recovery of precious metals, selenium and tellurium.

The wastes associated with selenium and tellurium recovery will be minor constituents of the total wastes from copper electrolytic refiners. As shown in Figure 2 (pg. 11) previously for every 1000 kg (1 MT) of electrolytic copper produced 1.1 kg of slimes are produced. These slimes are in turn processed for selenium, tellurium and precious metals recovery. Thus any quantity of residual waste remaining would be small. These residuals in the form of water and sludges would be disposed of along with the other waste sludges from the electrolytic copper plant. In Section 1.0 of this report dealing with electrolytic copper refineries the use of lined lagoons has been stipulated for adequate environmental protection. This practice would insure adequate disposal of the residues from selenium and tellurium and precious metals recovery.

For the above reasons and lack of specific data from the plants processing electrolytic slimes for selenium and tellurium recovery, quantification of wastes and treatment and disposal costs are not contained in this report.

Production processes are briefly described here and possible types and sources of hazardous wastes are identified.

13.2 WASTE CHARACTERIZATION

13.2.1 Process Description

Selenium Smelting and Refining. More than 90 percent of the domestic output of selenium is as a byproduct of copper refining. The selenium is recovered from the anode mud or slime that falls from the anode during electrorefining of copper. It also is recovered from some lead refining operations.

All commercial processes may be considered as modifications or combinations of three fundamental methods: Smelting with soda ash, roasting with soda ash, and roasting with sulfuric acid. In the first two methods, the selenium in the anode slimes is converted to sodium selenate which is soluble in water. Tellurium is precipitated from the solution by neutralization with acid and recovered by filtration, and then elemental selenium is precipitated with sulfur dioxide. In the third process, the anode slimes are roasted with sulfuric acid. Selenium dioxide volatilizes and is recovered in a wet Cottrell precipitator as selenious acid from which red amorphous selenium is precipitated by the addition of sulfur dioxide. Subsequent treatment of the various products resulting from the initial operations of all three processes recovers additional amounts of selenium. Overall recovery of selenium in the slimes is 60 to 70 percent (Ref. 7).

Tellurium Smelting and Refining. Tellurium-refining methods are closely guarded trade secrets, and modifications which different companies make in the standard flowsheet are not disclosed. Processing begins with tellurium-bearing anode residues and slags from the refining of copper and lead.

The anode slime is leached in acid solution to remove copper and then fused with sodium bisulfate to convert residual copper to copper sulfate and to volatilize part of the selenium. The fused cake is granulated and leached with water to remove the copper sulfate. A subsequent leach with caustic soda solution extracts the tellurium. Residue from the double leach is refined in a Doré furnace for recovery of precious metals. The tellurium is recovered from the caustic solution by acidifying to precipitate a tellurium oxide sludge that is subsequently refined to metal.

About 0.17 pound of tellurium is produced in the United States for each ton of copper produced. Inasmuch as copper anodes contain about 0.4 pound of tellurium per ton, only about 50 percent of the contained tellurium is recovered in a marketable form (Ref. 7).

With respect to the possible land disposal of potentially hazardous waste from selenium and tellurium (and precious metals which are also recovered from anode slimes and muds), it is important to note that estimated recovery of selenium and tellurium from copper anode slimes is 60 to 70 and 50 percent, respectively. It may be expected that since recovery is only 50 to 70% significant concentration of these potentially hazardous metals, along with arsenic, copper and lead and other heavy metals, will end up in wastewater and disposal slags from specialty smelters which process copper anode slimes and mud. A study of the specialty smelting plants listed in Table 99 would be needed to determine the exact quantities and disposition of potentially hazardous residuals from selenium and tellurium production.

14.0 PRIMARY SMELTING AND REFINING OF GOLD AND SILVER (SIC 3339)

14.1 INDUSTRY CHARACTERIZATION

Silver

Less than one percent of the new silver produced in the U.S. comes from simple silver of gold-silver ores amenable to direct processing by amalgamation and/or cyanidation. That is, practically all primary production is from ores bearing lead, zinc, and/or copper; silver-bearing concentrates floated from these ores are smelted at the facilities designed to produce primary lead, zinc, and copper as discussed elsewhere in this report.

A total of 36,494,000 troy oz. of silver were produced in 1973 (Ref. 2) at an average selling price of $2.558 per troy ounce. National sales of primary silver can then be estimated as $93,352,000.

The one percent share alluded to above amounts to roughly 200,000 troy ounces of silver and is produced at installations which are primarily gold smelters because the lode gold-silver ores from which the silver is produced contain much more gold than silver. The largest of these smelters is discussed below.

Gold

The 1973 U.S. production of gold metal was 1,322,000 troy ounces (Ref. 2). The average selling price of gold was $97.81 per troy ounce. The national sales for 1973 are estimated as 129,305,000.

One installation in South Dakota produces approximately 500,000 ounces (15.5 MT) of gold per year or about one-third of total U.S. production. Another one-third (approximately) is produced by two smelters in Nevada.

Gold and silver metal are produced by smelting a concentrate emanating from the ore treatment process as diagramed in Figure 20.

14.2 WASTE CHARACTERIZATION

Silver

As discussed previously nearly all of the U.S. primary silver production is from primary lead, zinc, and copper smelters. Negligible solid waste disposal is attributable to silver production at these smelters; the small amounts of solid residue generated are recycled completely to assure maximum recovery of silver.

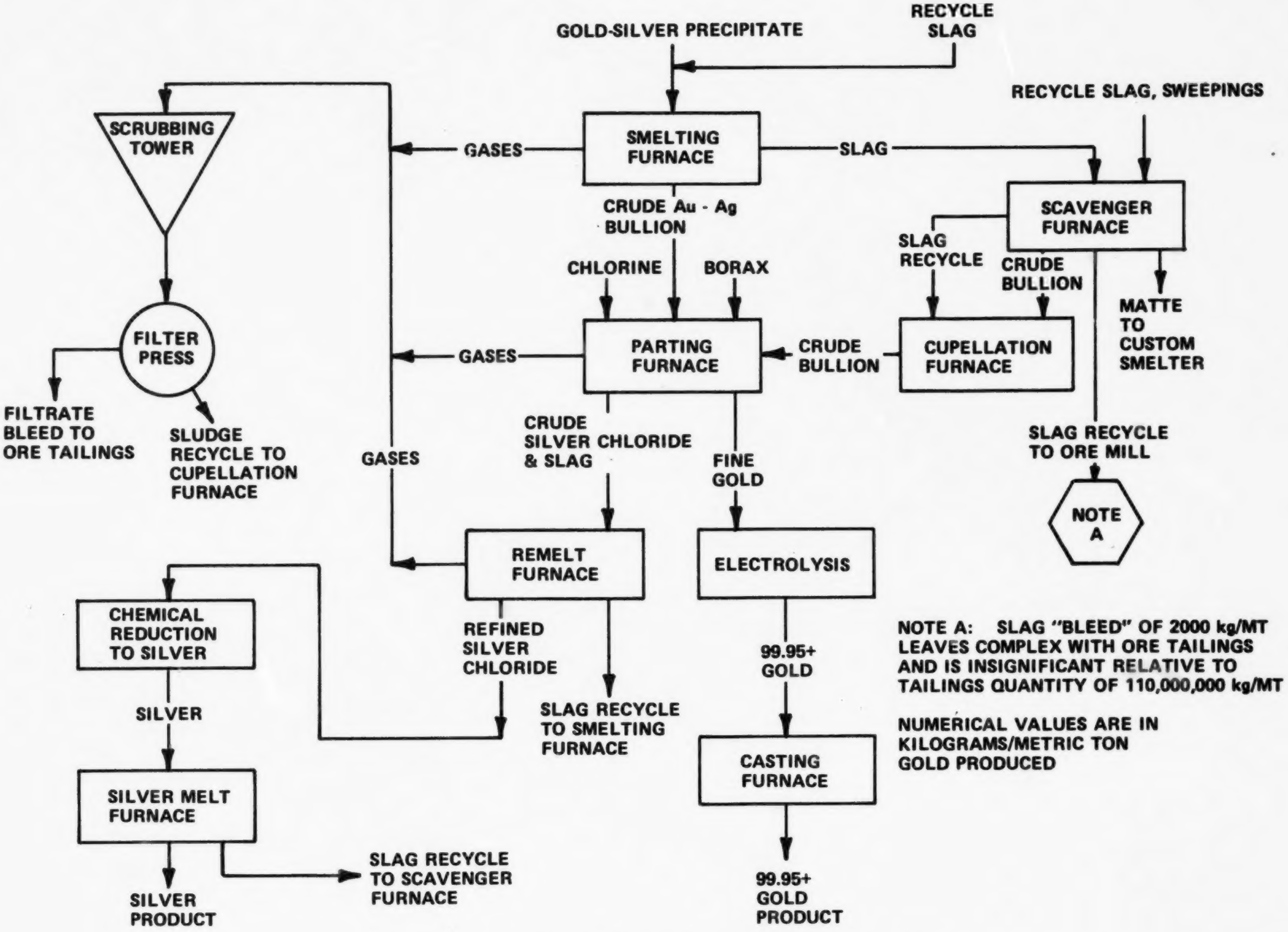

GOLD-SILVER PRECIPITATE
RECYCLE SLAG
RECYCLE SLAG, SWEEPINGS
SMELTING FURNACE
GASES
SLAG
SCAVENGER FURNACE
CRUDE Au - Ag BULLION
CHLORINE
BORAX
SLAG RECYCLE
CRUDE BULLION
MATTE TO CUSTOM SMELTER
SCRUBBING TOWER
FILTER PRESS
GASES
PARTING FURNACE
CRUDE BULLION
CUPELLATION FURNACE
FILTRATE BLEED TO ORE TAILINGS
SLUDGE RECYCLE TO CUPELLATION FURNACE
GASES
CRUDE SILVER CHLORIDE & SLAG
FINE GOLD
SLAG RECYCLE TO ORE MILL
NOTE A
REMELT FURNACE
ELECTROLYSIS
CHEMICAL REDUCTION TO SILVER
REFINED SILVER CHLORIDE
99.95+ GOLD
NOTE A: SLAG "BLEED" OF 2000 kg/MT LEAVES COMPLEX WITH ORE TAILINGS AND IS INSIGNIFICANT RELATIVE TO TAILINGS QUANTITY OF 110,000,000 kg/MT
SILVER
SLAG RECYCLE TO SMELTING FURNACE
CASTING FURNACE
NUMERICAL VALUES ARE IN KILOGRAMS/METRIC TON GOLD PRODUCED
SILVER MELT FURNACE
SLAG RECYCLE TO SCAVENGER FURNACE
SILVER PRODUCT
99.95+ GOLD PRODUCT

Figure 20 PRIMARY GOLD AND SILVER PRODUCTION

Gold

As shown in Figure 20 the only solid material not totally recycled or sold is that portion of the slag from the scavenger furnace which goes back to the ore treatment mill. This slag "bleed" amounting to less than 0.1 MT/day ultimately leaves the plant with the ore tailings and is insignificant compared to the mine tailings which amount to 4700 MT/day.

14.3 TREATMENT AND DISPOSAL TECHNOLOGY

Silver

As discussed previously primary silver is almost entirely recovered as by products of lead, zinc, and copper smelting or from gold ores. The minor amount of residues attributable to silver recovery at these primary smelters makes it unnecessary to describe Level I, II and III treatment and disposal technology.

Gold

As discussed previously, the only disposable waste from recovery of gold from concentrates is slag from the scavenger furnace. Less than 0.1 MT/day of this slag ultimately leaves the plant with ore tailings amounting to 4700 MT/day. Description of treatment and disposal technology for environmental protection is, therefore, not warranted.

14.4 COST ANALYSES

Since the quantities of land disposed wastes from primary silver and gold smelting and refining are negligible, costs of treatment and disposal technology are negligible.

15.0 PRIMARY SMELTING AND REFINING OF PLATINUM

15.1 INDUSTRY CHARACTERIZATION

As one would expect, the production of primary platinum in the United States is small. Production is reported in conjunction with palladium, iridium, osmium, rhodium and rutlemium which comprise the platinum-group metals. Production of the platinum-group metals from primary refiners ranged from 12,000 troy ounces to 21,000 troy ounces (0.373 MT to 0.653 MT) from 1968 to 1972 (Ref. 1). The major part of U.S. output is recovered as a specialty refining byproduct of copper refining in Maryland, New Jersey, Texas, Utah and Washington. Raw material for recovery of platinum-group metals are anode muds or slimes from electrolytic copper plants. Selenium, tellurium, gold and silver are also recovered from copper anode muds and slimes.

The wastes associated with platinum recovery will be minor constituents of the total wastes from copper electrolytic refiners. As shown in Figure 2 (page 11) previously, for every 1000 kg (1 MT) of electrolytic copper produced 1.1 kg of slimes are produced. These slimes are in turn processed for selenium, tellurium and precious metals recovery including platinum. Thus any quantity of residual waste remaining would be small. These residuals in the form of water and sludges would be disposed of along with the other waste sludges from the electrolytic copper plant. In Section 1.0 of this report dealing with electrolytic copper refineries the use of lined lagoons has been stipulated for adequate environmental protection. This practice would insure adequate disposal of the residues from platinum selenium and tellurium and other precious metals recovery from slimes.

For the above reasons and lack of specific data from the plants processing electrolytic slimes for platinum recovery, quantification of wastes and treatment and disposal costs are not contained in this report. Processes and possible sources of land disposed wastes with potentially hazardous constituents thereof are briefly described in the following paragraphs.

Copper anode muds containing platinum and other precious metals are washed and filtered to remove soluble copper which is recycled to copper electrolysis cells or disposed of to tailings impoundment if metal content is too low for recovery. Washed muds are roasted with sulfates and sulfuric acid, leached with water, and treated with copper powder to promote precious metal precipitation. Flue gases from the roaster are scrubbed (wet) and subjected to electrostatic precipitation to yield a recovered, marketable selenium and selenium oxide product and wastewater which is routed to disposal ponds.

The crude cemented precious metal concentrate, rich in copper, is isolated by filtration and the filtrate is routed to the copper electrowinning circuit. The concentrate is melted in special furnaces known as a Doré furnace with a variety of fluxing agents. Crude silver anodes and a slag which is routed to the copper smelter are derived from the Doré furnace. Fumes from the Doré furnace are routed through both wet and dry control systems and all solids are returned to the roaster while clarified solution is routed to waste ponds.

Doré metal anodes are electrorefined to recover silver, and anode muds are routed to a leaching circuit, induction furnace, and anode casting module for the preparation of gold anodes for electrorefining in Wöhlwill cells. Wastes generated in this circuit are either recycled to the silver recovery circuit (if solids) or to the platinum/palladium recovery circuit (in the case of spent or contaminated Wöhlwill cell electrolyte).

Spent Wöhlwill electrolyte is treated with ammonium chloride and chlorine to precipitate ammonium chloroplatinate which is isolated by filtration, dried, and reduced to platinum metal. Filtrate from the ammonium chloroplatinate isolation is transferred to the palladium recovery circuit and gases from the reduction furnace are vented to atmosphere through caustic scrubbers with scrubbing liquors going to wastewater impoundments.

As seen in the previous discussions, possible sources of land disposed wastes include: (1) filtrate from initial washing of copper anode muds, (2) roaster scrubwater which has been subjected to flotation for recovery of selenium, (3) clarifier water from Doré furnaces, and (4) scrubber liquor from reduction furnaces. It is presumed that these wastes are put into small lagoons. The volumes of liquid discharge to lagoons and resultant sludge quantities are not known but are relatively minor. Potentially hazardous constituents include As, Te, Se, Pb and Cu.

16.0 PRIMARY SMELTING AND REFINING OF BISMUTH

The quantity of primary bismuth metal produced in the United States is very small. Production figures are not divulged. Refined primary bismuth is produced only by American Smelting and Refining Company in Omaha, Nebraska (Ref. 1). Domestic production of approximately 381,000 lbs bismuth in 1973 was estimated by subtracting imports from exports and consumption (Ref. 3). The average 1973 selling price of bismuth was $4.92/lb. Thus national sales of domestic production for 1973 was estimated as $1,875,000.

As will be described the quantities of wastes associated with bismuth production are also expected to be small. In order to obtain accurate estimates of the quantities of total wastes and hazardous constituents thereof from bismuth smelting it would be necessary to visit and sample bismuth process waste streams from the Omaha plant. This was not done on this program. For these reasons bismuth refining was not studied in detail during this study.

Bismuth is always recovered as a byproduct of copper and lead smelting. In copper smelting, bismuth persists with copper until the converting operation, in which much of it volatilizes with lead, antimony, etc., as fumes. These are collected and sent to lead smelters. The portion of bismuth which escapes volatilization during converting is finally removed in the electrolytic copper refinery in the slimes. Treatment of these slimes for the recovery of gold and silver leaves bismuth associated with lead.

These slime residues are processed at lead or specialty smelters for recovery of bismuth and lead. Bismuth associated with lead ore accompanies lead through all steps of processing and is removed by special refining processes. When bismuth has been removed from lead by pyrometallurgical procedure, it is collected in a dross containing calcium, bismuth, magnesium, and lead. This dross is melted and chlorinated to remove calcium and magnesium as a molten chloride slag and a bismuth-lead alloy. The alloy is fluxed with molten sodium hydroxide to remove traces of arsenic and tellurium that may be present. Residual silver and gold are removed by treatment with zinc. The resultant lead-bismuth-zinc alloy is then subjected to chlorination treatment with chlorine gas under controlled conditions. Zinc and lead combine with chlorine preferentially over bismuth and are removed as molten chloride slags. The molten bismuth is finally treated with air and caustic soda and cast. This method of refining yields 99.999 per cent pure bismuth.

During the electrolytic refining of lead, bismuth enters the slimes with any arsenic, silver, gold, or antimony that may have been present. These slimes are filtered, dried, and melted to produce a slag and metal. The metal phase is blown with air to remove arsenic and antimony. It is then heated under oxidizing conditions, to produce a lead-bismuth slag and an alloy of gold and silver. The lead-bismuth slag is reduced by carbon to form a lead-bismuth alloy which is refined by chlorination as described above.

The only significant waste which may be land disposed is the calcium and magnesium chloride slag which may contain trace quantities of heavy metals. The quantity and exact disposition of this slag has not been ascertained because of no plant visits.

17.0 PRIMARY SMELTING AND REFINING OF COBALT

17.1 INDUSTRY CHARACTERIZATION

There is very little cobalt metal produced in the United States. U.S. production is reported by The Bureau of Mines (Ref. 1) as 146 MT in 1970 and 140 MT in 1971. There was not domestic smelter production of cobalt for 1972 or 1973 (Ref.2). There is only one U.S. smelter with the capabilities to produce cobalt metal. This is the American Metal Climax, Port Nickel, La. refinery. For these reasons the industry has not been investigated in detail during the conduct of this study.

A flow diagram and brief descriptions of potentially hazardous residuals which may be land disposed is presented here. A flow diagram for production of cobalt metal is shown in Figure 21. Iron pyrite concentrate, containing cobalt value and produced by flotation of ore, is roasted to convert sulfide ore minerals to oxides, sulfates, and similar compounds. While details of dust control in roaster off gases are not available, the use of these off gases in a sulfuric acid plant suggests that entrained dusts are removed and either recycled or disposed of. These dusts contain Cu, Co, Ni, and Zn and are therefore a potential hazard if land disposed.

Following roasting, the calcine is subjected to a chloridizing roast to produce soluble chloride salts. Emissions contain Cu, Co, Ni and Zn dust which pose a potential hazard if disposed of on land after collection. The chloridized calcine is leached with water, filtered and a filter cake washed to yield a chloride solution and a filter cake of iron oxide which serves as feed for the steel industry. The chloride solution is treated by scrap iron cementation (to remove copper), controlled neutralization (to precipitate iron values), and finally filtration to isolate a cobalt chloride solution. The cement copper is sent to copper refinery operations for recovery and the iron hydroxide is either sintered to provide steel industry feed or disposed of to landfill or pond impoundments. The iron hydroxide cake contains chlorides of Cu, Co, Ni, and Zn and must be considered potentially hazardous if land disposed.

The cobalt chloride solution is treated with sodium carbonate and chlorine gas to produce an insoluble cobalt hydrous oxide which is subsequently recovered by filtration. The filtrate, containing nickel, zinc, chloride, and other species is apparently discharged as waste to tailings disposal lagoons and thus must be considered potentially hazardous. The hydrous cobalt oxide is next dried and then reduced, using a charcoal reductant, to produce cobalt metal.

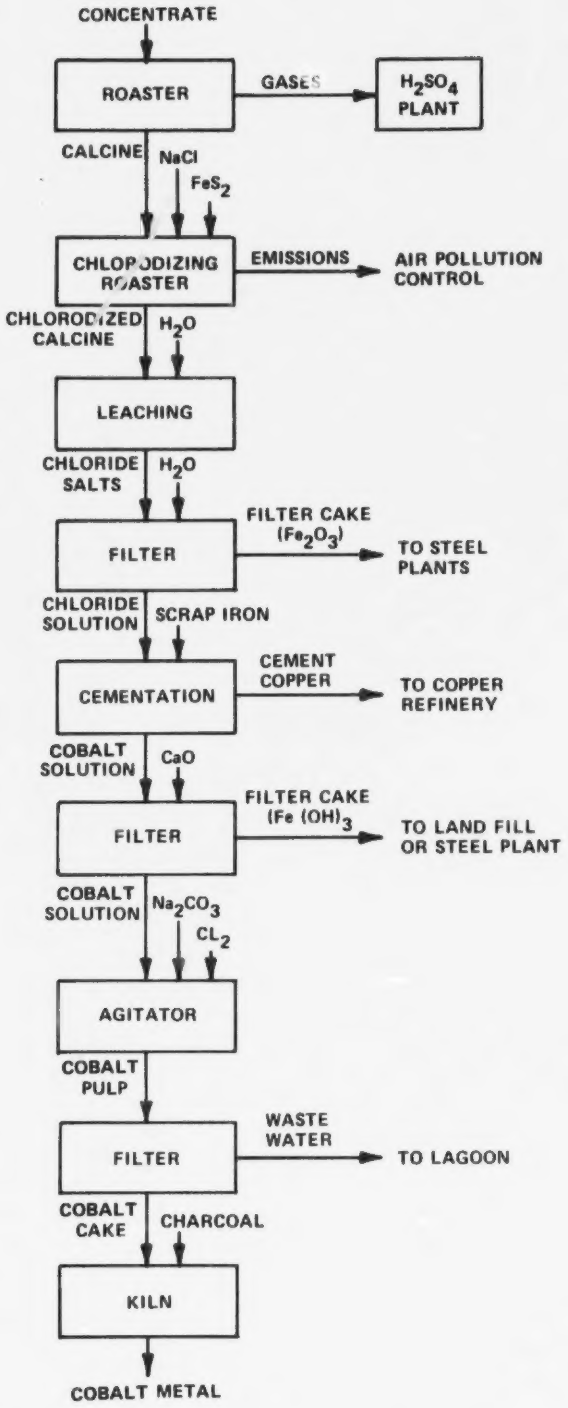

CONCENTRATE
ROASTER
GASES
H_2SO_4 PLANT
CALCINE
NaCl
FeS_2
CHLORODIZING ROASTER
EMISSIONS
AIR POLLUTION CONTROL
CHLORODIZED CALCINE
H_2O
LEACHING
CHLORIDE SALTS
H_2O
FILTER
FILTER CAKE (Fe_2O_3)
TO STEEL PLANTS
CHLORIDE SOLUTION
SCRAP IRON
CEMENTATION
CEMENT COPPER
TO COPPER REFINERY
COBALT SOLUTION
CaO
FILTER
FILTER CAKE ($Fe(OH)_3$)
TO LAND FILL OR STEEL PLANT
COBALT SOLUTION
Na_2CO_3
CL_2
AGITATOR
COBALT PULP
FILTER
WASTE WATER
TO LAGOON
COBALT CAKE
CHARCOAL
KILN
COBALT METAL

Figure 21 PRIMARY COBALT SMELTING AND REFINING

18.0 PRIMARY SMELTING AND REFINING OF ZIRCONIUM AND HAFNIUM

Only small amounts of zirconium and hafnium sponge metal are presently produced in the United States. The only U.S. producers of these metals in 1974 were the Wah Chang, Albany, Oregon, Division of Teledyne Corporation and the Amax Specialty Metals Division of Amax, Inc., Akron, New York (Ref. 9). The New York plant closed in 1975. Statistical data on production of zirconium sponge and metal and hafnium sponge is withheld to avoid disclosure. Production of hafnium crystal bar which is made from hafnium sponge was reported as only 36 MT in 1972 (Ref. 1). U.S. consumption of zirconium metal in 1973 was 2722 MT (Ref. 2).

Zirconium and hafnium are used in nuclear reactors and production of these metals is expected to increase if there is expansion of nuclear power in the United States.

In order to obtain estimates of the types and quantities of solid and hazardous wastes from the smelting and refining of zirconium and hafnium metal it will be necessary to visit and sample the Oregon plant. This was not done under the current program. For this reason, and because of minor production, detailed presentations on solid and hazardous waste generation, treatment and disposal technology and associated costs are not given in this report. Production processes are discussed and possible sources of potentially hazardous wastes identified.

Production of Zirconium and Hafnium. The elements zirconium and hafnium always occur together in minerals which are associated in sand deposits with minerals containing titanium and rare-earth elements. The earth's crust has been estimated to contain approximately 50 ppm of zirconium minerals. Zircon ($ZrSiO_4$) and baddeleyite (ZrO_2), are marketed commercially, but zircon is by far the more important source (Ref. 7).

Two types of zirconium are produced - commercial grade, and reactor grade to be used in nuclear reactors. Commercial-grade zirconium is produced by blending zircon with coke which is then reacted or fused in an electric arc furnace. The carbon and the nitrogen from air react with zirconium to yield zirconium carbonitride containing about 85 percent zirconium. The carbonitride is chlorinated to crude zirconium tetrachloride which is reduced to zirconium metal with magnesium metal under an inert atmoshpere. The reduction stage is called the Kroll process. The resulting mixture of zirconium metal and magnesium chloride is vacuum-distilled to remove the magnesium chloride. The residual zirconium sponge contains about 2 percent hafnium but is still suitable for most nonnuclear applications. Direct chlorination of zircon concentrates to produce the mixed chlorides also is reportedly utilized commercially.

To produce reactor-grade or hafnium-free zirconium, crude zirconium tetrachloride is dissolved in acid containing ammonium thiocyanate complexing agent and put through solvent extraction, which strips hafnium from the solution. The zirconium component is precipitated and calcined to oxide which is rechlorinated to high-purity zirconium tetrachloride and reduced by the Kroll process.

The resulting zirconium sponge is crushed, compacted into consumable electrodes, and vacuum melted in an inert atmosphere into ingot. The ingot is used as consumable electrode to produce second-melt ingot, which is machined and readied for fabrication.

The solvent used to remove hafnium from zirconium is methyl isobutyl ketone (MIBK). The solvent is lighter than and is not miscible with the aqueous solution. Hafnium thiocyanate dissolves preferentially in the solvent as it passes upward through the column. The hafnium thiocyanate is stripped from the organic phase with sulfuric acid, at which time it is converted to the sulfate. Hafnium is precipitated as the hydroxide and is then roasted to the oxide containing about 99 percent hafnium oxide.

Hafnium oxide, mixed with carbon, surcrose, and a small amount of water, is briquetted, dried, and then chlorinated in a vertical-shaft chlorinator. The resulting hafnium chloride vapors are purified by injection into a molten mixture of potassium chloride (KCl) and sodium chloride (NaCl), containing a small quantity of metallic sodium. High-purity hafnium chloride is volatilized from the salt bath and is reacted with magnesium in a Kroll-type reaction at about 800° C. When the furnace has cooled, the resulting mixture of magnesium and sodium chlorides and any excess magnesium is volatilized, and the residual hafnium sponge is removed from the crucible.

Further refining of the sponge metal may be accomplished by the iodide process, by electrorefining, or by electron beam melting.

Sources of potentially hazardous wastes include dilute filter solutions containing ammonium thiocyanate (NH_4CNO) and methyl isobutyl ketone (MIBK).

SECTION III

SECONDARY NON-FERROUS SMELTING AND REFINING

This section presents the results of investigations and analyses of on-land disposal or storage of process and pollution control residuals from the United States secondary nonferrous smelting and refining industry. Characteristics of each industry sector including plant locations, production capacities and smelting and refining processes have been identified and described.

Land disposed or stored residuals including slags, dusts, and sludges have been identified and characterized physically and chemically. State, regional and national estimates have been made of the total quantities of land disposed or stored residuals and potentially hazardous constituents thereof.

Current methods employed by the secondary metals industry for the disposal or storage of process and pollution control residuals on land are described. Principal methods include lagoon storage of sludges and open dumping of slags. Methods of residual treatment and disposal considered suitable for adequate health and environmental protection have been described. Finally, the costs incurred by typical plants in each secondary smelting and refining category for current and environmentally sound residual disposal or storage on land, have been estimated.

Secondary metals are categorized to include all metals and metal alloys recovered from scrap and waste origins. Final products are produced with chemical composition specifications derived according to individual materials application requirements. Individual alloy products are indistinguishable in quality from comparable grade materials made by primary producers. The feasibility of producing specific grades metal alloys at secondary smelters is, therefore, governed mainly by cost competition with primary producers. Some of the techniques employed at secondary smelters to meet alloy specifications often include the addition of primary metals for dilution or alloying purposes. On the other hand, much of the waste products generated at the secondary smelter often contain metal values exceeding those present in the corresponding ore concentrates and, thus presents a raw materials source for primary smelting.

The difficulty in characterizing solid waste generated from secondary metals smelting by SIC categorization lies in the fact that secondary metals are produced mainly according to the nature of scrap supply sources rather than specific SIC metal classifications. The extent of waste produced from secondary smelting of each metal category will differ greatly according to the metallurgical processing required. The amount of waste requiring direct land disposal by the smelter will also depend on whether there are demands for such waste products for further metal recovery or alternate applications. Thus, an assessment of prevalent operations within the individual secondary metals SIC classifications is necessary so that pertinent subcategorization of hazardous waste sectors within the SIC 3341 secondary metals smelting industry can be derived.

SIC 33412 -- Secondary Smelting and Refining of Copper

As shown by the 1971 and 1972 production data summarized in the U.S. Bureau of Mines-Minerals Yearbook (Reference 1), unalloyed copper production at secondary smelters accounts for only about 15 percent of such a product from scrap sources. Further, the refined copper produced by secondary smelters is accounted for almost entirely by recovery from new scraps. In the case of secondary copper recovered as alloy products, brass mills account for about 65 percent of the total production. Brass mill products, however, are generated mainly from new scraps by melting of appropriately formulated alloy heats. Waste dross generated from the flux cover used in the melting process is shipped to primary or secondary smelters for further metal recovery. Land-destined wastes within the SIC 33412 category are, therefore, attributed mainly to the production of alloy products from old scraps at secondary smelters. In addition, it should be noted that significant quantities of tin, lead, zinc, and nickel are recovered in the production of secondary copper alloys. About 47 percent of the secondary tin recovered is in the remelting of brass and bronze scraps.

SIC 33413--Secondary Smelting and Refining of Lead

Secondary lead recovery is one of the oldest known recycling operations. Today, the volume of secondary lead recovery represents nearly 50 percent of the total lead production in this country. About 29 percent of the lead produced from secondary sources is in the form of refined pure lead, the remainder is recovered in a variety of different lead alloys. Antimonial lead used in storage battery plates represents by far the single most important product of the secondary lead smelting industry, accounting for 57 percent of the total secondary lead recovery. Another category of lead alloys, consisting of varying mixtures of lead, tin, and antimony, are known as white metals. Alloy scraps of these types are generally grouped together for making up specified heats in the production of various type metal, babbitt, and solder alloys. These white metal alloys account for about 14 percent of the secondary lead market. Thus, for the purpose of assessing land-destined wastes within the SIC 33413 category, three different types of processing operations, corresponding to soft lead, antimonial lead, and white metal productions, must be considered. Secondary antimony recovered with lead alloys represents not only the entire secondary antimony production but also about 63 percent of the total antimony production in this country.

SIC 33413 -- Secondary Smelting and Refining of Zinc

By far the largest single smelting operation for secondary zinc recovery is in the remelting of copper base alloys, accounting for 46 percent of the secondary zinc recycle and 53 percent of metal recovery. About 13 percent of the scrap zinc recycle is in the form of metal value recovery as zinc chemicals. Zinc recovery by remelting the various zinc-containing alloy scraps accounts for about 54 percent of the secondary zinc smelting operation. Scrap distillation for recovery of pure zinc in the form of either slab zinc or zinc dust accounts for the remaining 33 percent of the secondary zinc production market. The secondary zinc smelting industry is beset with air pollution control problems, but very little land-destined wastes are generated. Dust residues collected from air pollution control devices generally have very high zinc contents and are used in zinc chemical productions principally zinc oxide. This accounts for the fact that a rather high percent of the secondary zinc recovery market is in the production of zinc chemicals. In view of the fact that little solid wastes are disposed within the SIC 33414 industry sector, a detailed evaluation of hazardous waste generation in this industry category is not provided in this study.

SIC 33415 -- Secondary Smelting and Refining of Antimony, Cadmium, Magnesium, Nickel, Tin and Titanium

As noted early in the assessment of copper- and lead-based alloy smelting operations covered respectively under SIC 33412 and 33413, secondary antimony is recovered entirely with lead base alloys and secondary tin is recovered to the extent of 90 percent as either lead-based or copper-based alloys. The remaining 10 percent of secondary tin production is ascribable to detinning operations which are not normally classified under the secondary metals smelting industry category.

Cadmium is widely used in plating applications and secondary recovery of this metal is found to be practically nonexistent as a separate industry. Titanium is a refractory metal gaining popularity as a structural material only in recent years. Secondary titanium smelting is also found to be essentially nonexistent.

There is no distinct secondary nickel smelting and refining industry. Scrap containing nickel is routed through scrap dealers to the following consumers: (Ref.2)

nonferrous smelters and refiners
steel mills
foundries
alloy producers

Almost invariably stainless steel scrap containing nickel is used to make stainless steel, nickel-bearing alloy scrap is used to make nickel-bearing alloys. Any wastes associated with the processing of nickel-bearing scrap will occur at the aforementioned plants and be included in studies of these smelting and refining sectors.

Secondary magnesium is recovered only to a minor extent in about six plants in the United States. Secondary smelting of magnesium is, in addition, limited only to remelting, dilution and alloying operations. Very little land-destined wastes are generated. Hazardous constituents are not found in these wastes.

On the basis of the above considerations, quantitative data on hazardous waste generation and disposal in the SIC 33416 category were, therefore, not derived as a separate part of this study.

SIC 33417 -- Secondary Smelting and Refining of Aluminum

Secondary aluminum recycle is one of the most important segments of the secondary nonferrous metals smelting industry category. Although a large variety of aluminum alloys are produced by secondary aluminum smelters, most of the alloys contain only minor amounts of alloying elements. Thus, the secondary aluminum smelter is in direct competition with the primary producer far more closely in product lines in comparison with other nonferrous metals smelting categories. Aluminum is an easily oxidizable metal. Only the removal of magnesium and oxide impurities can be accomplished in secondary aluminum smelting. A majority of secondary aluminum smelters are engaged only in remelting and demagging (i.e. removal of magnesium) of high grade aluminum scraps. A number of smelters do have capabilities for aluminum recovery from dross waste generated in remelting and foundry operations. Waste generation and disposal problems faced by high grade scrap vs. dross smelters are far different. Separate assessments for each of these categories are, therefore provided in this study.

On the basis of prevalent secondary nonferrous metals smelting practices within the recycling industry, detailed hazardous waste assessments and data compilations are provided in the following categories:

Secondary Copper Smelting (SIC 33412, 33414, 33416)
Secondary Soft Lead Smelting (SIC 33413)
Secondary Antimonial Lead Smelting (SIC 33413, 33416)
Secondary White Metal Smelting (SIC 33413, 33416)
Secondary Scrap Aluminum Smelting (SIC 33417)
Secondary Aluminum Dross Smelting (SIC 33417)

1.0 SECONDARY SMELTING AND REFINING OF COPPER (SIC 33412)

1.1 INDUSTRY CHARACTERIZATION

Recovery of copper from scrap is not conducted entirely at secondary copper smelters. Of 1.18 million MT of copper recovered in 1972 28% was from primary producers, 44% from brass mills and 23% from secondary smelters (Ref. 1). The remaining 5% was reclaimed at chemcial plants, foundries and manufacturers. Solid wastes from primary smelters is handled elsewhere in this report. Brass mills engage in scrap melting only and as a result produce negligible waste.

There are an estimated 40 secondary copper smelters in the United States. Plant capacities vary widely from 500 MT per year to about 80,000 MT per year. The typical pyrometallurgical plant produces 10,000 MT of copper alloy metals per year and operates 250 to 300 days/year. This represents the median value taken from the plant size distribution of pyrometallurgical plants. Plants employing electrolytic refining typically produce 40,000 MT of copper per year although the smallest electrolytic plant produces only 540 MT copper per year and the largest approximately 80,000 MT per year. Table 100 gives the estimated state by state capacities for secondary copper smelters. Production capacity is missing for 4 plants. According to the 1972 Minerals Yearbook 255,431 MT of copper was produced at secondary copper smelters in 1971 and 266,224 MT in 1972 (Ref. 1). Twenty-two percent of the 1.36 million tons of scrap copper or 267,443 MT was recovered at secondary copper smelters in 1973 (Ref. 2). The average selling price of copper in 1973 was 59.5 cents/lb. Thus the national sales of copper from secondary smelters may be estimated as $350,083,000.

1.2 WASTE CHARACTERIZATION

This section contains descriptions of production technology at secondary copper smelters and the resultant by products or wastes which are either recycled directly, shipped to other smelters for further metallic recovery or disposed of on land. Estimates are given for the quantities of wastes and potentially hazardous constituents which are disposed of on land either in lagoons or open dumps.

1.2.1 Process Descriptions

Pyrometallurgical. A generalized secondary copper and brass smelting process waste flow diagram is shown in Figure 22. The typical secondary copper producer will engage in the production of copper and brass and bronze alloys by pyrometallurgical processes. Electrolytically refined copper is more likely to be produced at the primary copper refiner although 4 secondary plants produce electrolytic copper. The major source of solid wastes in the secondary copper smelter is in the blast furnace recovery of copper.

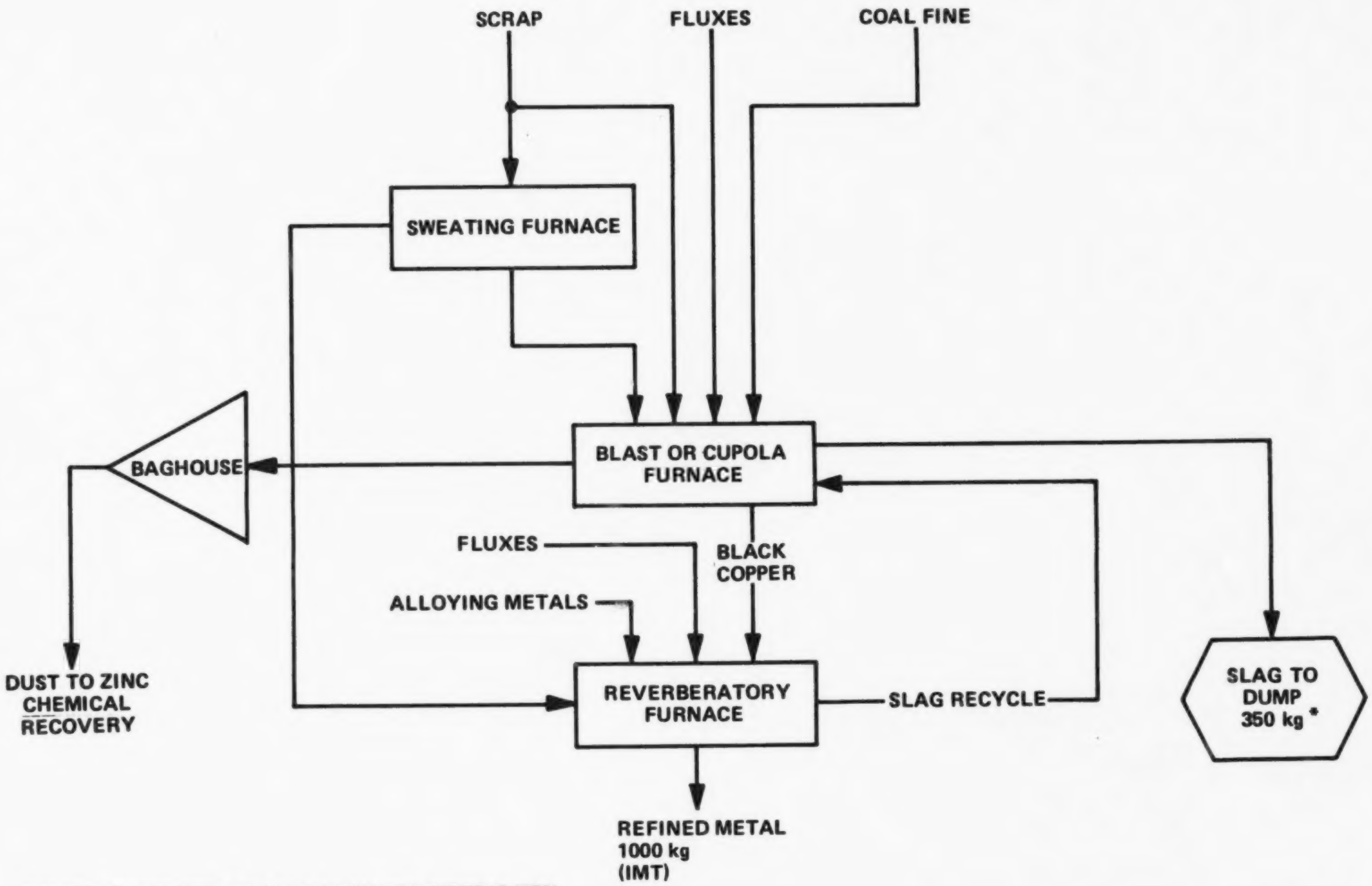
Table 100
GEOGRAPHICAL DISTRIBUTION OF SECONDARY COPPER SMELTERS

STATE	NO.OF PLANTS	PLANT DISTRIBUTION BY CAPACITY (MT)			PLANT DISTRIBUTION BY PROCESS		CAPACITY PER PROCESS (MT)	
		2 - 5K	5 - 15K	15 - 40K	P	E	P	E
ALABAMA	2	1	1	0	2	0	12,480	0
CALIFORNIA *	2	0	1	0	2	0	5,940	0
CONNECTICUT	1	1	0	0	1	0	1,080	0
DELAWARE	1	0	0	1	1	0	16,320	0
GEORGIA	1	0	0	1	0	1	0	32,640
ILLINOIS *	7	1	3	3	5	2	72,400	114,300
INDIANA	3	2	1	0	3	0	13,380	0
KANSAS	1	0	1	0	1	0	7,620	0
MASSACHUSETTS	2	1	1	0	2	0	9,750	0
MICHIGAN	1	0	1	0	1	0	6,840	0
NEW JERSEY *	3	0	1	0	3	0	8,000	0
NEW YORK	3	2	1	0	3	0	49,830	0
OHIO	4	1	3	0	4	0	16,860	0
PENNSYLVANIA *	7	1	2	3	7	0	76,920	0
TEXAS	1	1	0	0	1	0	4,900	0
UTAH	1	1	0	0	0	1	0	540
EPA REGION								
I *	3	2	1	0	3	0	10,830	0
II *	6	2	2	0	6	0	57,830	0
III *	8	1	2	4	8	0	93,240	0
IV	3	1	1	1	2	1	12,480	32,640
V *	15	4	8	3	13	2	109,480	114,300
VI	1	1	0	0	1	0	4,900	0
VII *	1	0	1	0	1	0	7,620	0
VIII	1	1	0	0	0	1	0	540
IX *	2	0	1	0	2	0	5,940	0
X	0	0	0	0	0	0	0	0
NATION *	40	12	16	8	36	4	302,320	147,480

PRODUCTION/CAPACITY DATA MISSING FOR 4 PLANTS

NOTE: *AT LEAST ONE PLANT HAD NO CAPACITY LEVEL GIVEN

P – PYROMETALLURGICAL
E – ELECTROLYTIC


SCRAP
FLUXES
COAL FINE
SWEATING FURNACE
BLAST OR CUPOLA FURNACE
BAGHOUSE
DUST TO ZINC CHEMICAL RECOVERY
FLUXES
ALLOYING METALS
BLACK COPPER
REVERBERATORY FURNACE
SLAG RECYCLE
SLAG TO DUMP 350 kg *
REFINED METAL 1000 kg (IMT)

*WASTE QUANTITIES IN KILOGRAM PER METRIC TON REFINED METAL DRY WEIGHT

Figure 22 SECONDARY COPPER SMELTING AND PYROMETALLURGICAL REFINING

It is important to note that when high grade scrap is smelted the melted scrap is sent directly to the reverbatory furnace rather than to the blast furnace. In this case the slag is not disposed but recycled for metal recovery in the blast furnace.

Blast furnace processing in the secondary copper smelting plant is mainly intended for recycle of drosses, skimmings and slag waste accumulated from high grade scrap melting and smelting operations. The blast furnace may be operated periodically only when sufficient waste byproducts are accumulated for at least one week continuous furnace operation. Only about 20 secondary copper smelting plants in the United States have blast furnace scrap processing capabilities.

During cupola and blast furnace processing, low grade scrap mixtures are melted with suitable flux chemicals, such as limestone, borax and waste glass. Coke is charged for reduction of copper compounds to the metal. The blast air is blown in through tuyeres near the bottom of the furnace. Each heat of scrap and flux is maintained for about a day. The molten metal and slag are tapped intermittently. A basic difference between cupolas and blast furnaces lies in the fact that the former are not intended for copper compound reduction and so less fuel and lower blast temperatures are used. The intermediate metal product recovered is known as black copper. With blast furnace charges from typical scrap sources, average composition of the metal is as follows: Cu, 83.7%; Sn, 3.7%; Pb, 3.8%; Sb, 0.2%; Fe, 2.5%; Ni, 0.8%; S, 0.5%, Zn, 4.8%. Black copper is further refined in converter and anode furnaces where the impurity metal constituents are converted to oxides and removed with the added flux as slag. The slag is, in turn, recycled to the blast furnace for further copper recovery.

The "typical" plant may be assumed to have an annual total copper alloy production volume of 10,000 metric tons (MT). Only about 1,500 MT black copper is produced annually during the overall smelting process cycle. This black copper is consumed in-plant for brass and bronze alloy production. The typical blast furnace daily production capacity may be assumed to be 15 MT.

Electrolytic Refining. The principal difference between secondary copper plants using strictly pyrometallurgical processes for refining and those using the electrolytic refining is that the copper from the reverbatory furnace is further refined by first dissolving in sulfuric and then electrolytically redepositing pure copper on the cathodes of electrolytic cells. This process is shown as Figure 23. The typical electrolytic secondary copper plant is larger than the plants which strictly employ pyrometallurgical processes and will produce about 40,000 MT of copper per year. One electrolytic plant produces about 80,000 MT copper per year.

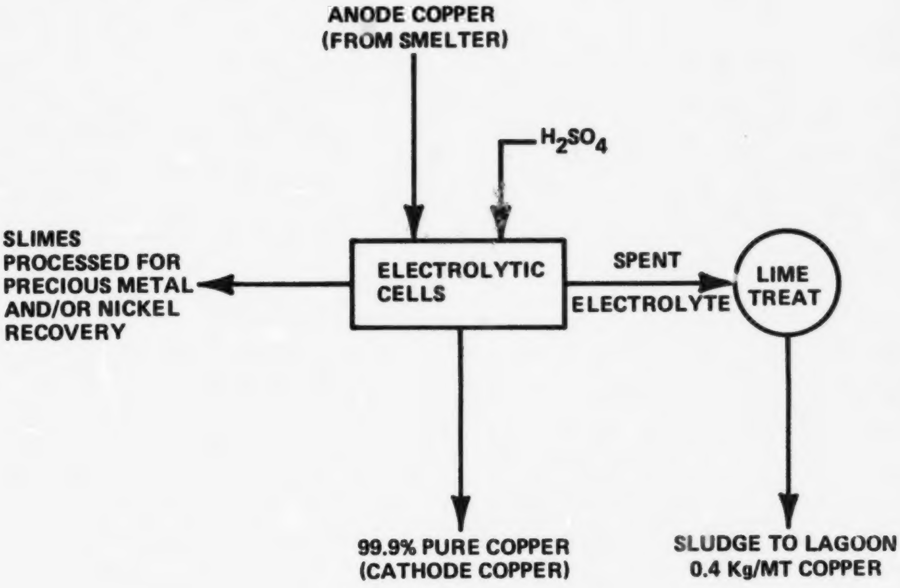

ANODE COPPER (FROM SMELTER)
H_2SO_4
SLIMES PROCESSED FOR PRECIOUS METAL AND/OR NICKEL RECOVERY
ELECTROLYTIC CELLS
SPENT ELECTROLYTE
LIME TREAT
99.9% PURE COPPER (CATHODE COPPER)
SLUDGE TO LAGOON 0.4 Kg/MT COPPER

Figure 23 SECONDARY COPPER ELECTROLYTIC REFINING

Description of Waste Streams. The typical blast furnace dust will have the following chemical composition: Zn, 58-61%, Pb, 2-8%; Sn, 5-15%; Cu, 0.5%; Sb, 0.1%; and Cl, 0.1-0.5%. Because of the high zinc content, the dust can be recycled for zinc compound manufacture largely as paint additives. This waste byproduct is thus sold to chemical and paint manufacturers and does not require direct disposal by the smelter.

Pyrometallurgical. The typical blast furnace slag will have the following composition: FeO, 15-20%; CaO, 7-11%; SiO_2, 30-34%; Al_2O_3, 4-6%; Cu, 1.6%; Sn, 0.7%; Pb, 0.3%; Zn, 1%. Analyses of samples obtained on this study are given in Appendix A. At the assumed "typical" plant, about 3,500 MT blast furnace slag is produced annually. During the period of blast furnace operation, this slag waste is produced at the rate of 35 MT per day. This waste is disposed by either on-site or off-site by open dumping. The slag material is normally tapped and cast in crucibles at about 100 kg sizes. This material is present mainly in a silicate type chemical structure with high density, hardness and impact strength. Although this slag would not be expected to easily leach solubility tests as described in Appendix B showed significant solubility of zinc, cadmium, copper, and lead. For this reason blast furnace slag from secondary copper smelters must be considered potentially hazardous at this time.

Electrolytic. Lime treatment of spent electrolyte generates a predominantly lime sludge containing significant concentrations of nickel, zinc, copper, chromium and cadmium. This sludge is generated at a rate of approximately 1.3 Kg wet sludge/MT copper or approximately 0.4 Kg. dry weight per MT copper. Appendix A contains chemical analyses of residuals from secondary copper smelters.

1.2.3 Waste Quantities. Table 101 summarizes the generation factors for the land disposed residuals from the secondary copper smelting and refining category (i.e. blast furnace slag, electrolyte sludge). This table also gives typical concentrations of potentially hazardous constituents. Table 102 gives the yearly quantity of wastes and hazardous constituents thereof for a typical pyrometallurgical plant producing 10,000 MT of brass and bronze ingot or anode copper per year. Table 102 gives these values for a typical secondary electrolytic copper plant producing 40,000 MT of electrolytic (i.e. cathode) copper.

Using the waste generation factors and concentration factors contained in Table 101 and and information on the geographic distribution of smelter types, capacities and pollution control methods estimates have been made for the quantities of land disposed or stored wastes and hazardous constituents thereof from secondary copper smelting and refining. These estimates are given for 1974, 1977 and 1983 on a state-by-state, regional and national level in Table 103.

TABLE 101

WASTE RESIDUAL FACTORS, SECONDARY COPPER SMELTING AND REFINING (DRY WEIGHT)

Smelter Category, and Type of Waste	Waste Generation Factor (kg/MT Production)	Concentration Factor, ppm								
		Cd	Cr	Cu	Mn	Ni	Pb	Sb	Sn	Zn
Blast Furnace Slag	350	5	20	16,000	7,000	260	3,000	100	7,000	10,000
Electrolyte Wastewater Sludge	0.4	NOT KNOWN								

* Mulitply sludge factor by 3 to obtain approximate wet weight.

TABLE 102

Yearly Generation of Waste Residuals From Typical Plant
Secondary Copper Smelting and Refining (Dry Weight)

Type of Waste	Total Waste Quantity (MT)	Quantity of Potentially Hazardous Constituents (MT) Pb	Cu	Zn	Ni	Cr	Mn	Sn
Pyrometallurgical								
Blast furnace slag	3500	1	56	35	0.1	.01	24	24
Electrolytic								
Electrolytic wastewater sludge	16	NOT KNOWN						
Blast furnace slag	14,000	4	224	140	0.4	.04	96	96

* Quantities calculated from generation and concentration factors given in Table 101 based on annual production of 10,000 MT pyrometallurgical and 40,000 electrolytic. Divide by 250 to obtain daily quantities. Multiply by 1.1 to convert to short tons. Multiply sludge no. by 3.0 to obtain approximate wet weight.

Table 103-a
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY COPPER AND BRASS AND BRONZE SMELTING TOTAL BLAST FURNANCE SLAG - 1974, 1977, 1983 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS					
					Cu	Mn	Ni	Pb	Sn	Zn
CALIFORNIA	6,300	6,300	273	OPEN DUMP	101	44	2	19	44	63
GEORGIA	38,000	38,000	1644		608	266	10	114	266	380
ILLINOIS	37,500	37,500	1623		600	262	10	113	263	375
INDIANA	2,800	2,800	122		45	20	1	8	20	28
KANSAS	2,500	2,500	109		40	18	1	7	18	25
NEW JERSEY	14,000	14,000	606		224	98	4	42	98	140
NEW YORK	4,900	4,900	211		78	34	1	15	34	49
OHIO	3,400	3,400	147		54	24	1	10	24	34
PENNSYLVANIA	43,900	43,900	1900		702	307	13	132	307	439
EPA REGION II	18,900	18,900	817		302	132	5	57	132	189
III	43,900	43,900	1900		702	307	13	132	307	439
IV	38,000	38,000	1644		608	266	10	114	266	380
V	43,700	43,700	1892		699	306	12	131	307	437
VII	2,500	2,500	109		40	18	1	7	18	25
IX	6,300	6,300	273		101	44	2	19	44	63
NATIONAL TOTAL	153,300	153,300	6635		2452	1074	43	460	1074	1533

*MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

Table 103-b

ESTIMATED STATE, REGIONAL AND NATIONAL WASTE FROM SECONDARY COPPER ELECTROLYTIC REFINING (CATHODE COPPER) TOTAL ELECTROLYTE WASTE WATER SLUDGE, 1974, 1977, 1983 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS					
					Cu	Mn	Ni	Pb	Sn	Zn
GEORGIA	13	13	NOT KNOWN	LAGOON	NOT KNOWN					
ILLINOIS	46	46	NOT KNOWN		NOT KNOWN					
UTAH	0.2	0.2	NOT KNOWN		NOT KNOWN					
EPA REGION										
IV	13	13	NOT KNOWN		NOT KNOWN					
V	46	46	NOT KNOWN		NOT KNOWN					
VIII	0.2	0.2	NOT KNOWN		NOT KNOWN					
NATIONAL	59.2	59.2	NOT KNOWN		NOT KNOWN					

*MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORP.

Principal products of secondary copper smelters are copper-base alloys in the form of brass and bronze ingots. Production figures for secondary copper smelters are therefore most appropriately represented by the annual statistics for brass and bronze ingot deliveries as compiled by the American Bureau of Metal Statistic (ABMS). Brass and bronze produced from secondary copper recycle and reused in captive brass mills are not included among these delivery figures. Over the past eleven-year period from 1963 to 1973, annual delivery volumes showed only slight variations with the average value remaining remarkably constant at about 300,000 MT per year. It may thus be assumed that the secondary copper smelting product volumes in the years 1977 and 1983 will not be significantly different from current production figures. Significant changes in secondary copper smelting process technology are not expected for the short term. Hazardous waste data for the years 1977 and 1983 are therefore projected to be essentially the same as those for 1974. States producing the largest quantity of potentially hazardous slag waste are Georgia, Illinois and Pennsylvania. Sludges are generated only from electrolytic refiners and are of significance only in Georgia and Illinois

1.3 TREATMENT AND DISPOSAL TECHNOLOGY

1.3.1 Current Waste Treatment And Disposal Practices

Currently slags from pyrometallurgical and electrolytic plants are open dumped. Electrolytic plant wastewater sludges is settled in unlined lagoons. As previously discussed, these wastes are considered potentially hazardous at the present time. Levels of treatment and technology and environmental adequacy thereof for these wastes are described in the following sections.

1.3.2 Present Treatment and Disposal Technology (Level I)

Pyrometallurgical. At the present time blast furnace slags generated at a rate of 350 kg/MT of copper production are open dumped. This practice is inadequate if leached heavy metals including zinc, copper and lead percolate through permeable soil to groundwater.

Electrolytic. Spent electrolyte wastewater treatment sludge generated at a rate of 0.4 kg/MT copper (1.2 kg/MT wet weight) is presently disposed of in unlined lagoons. This practice is inadequate if heavy metals including zinc, copper and lead percolate through permeable soils to groundwater. Electrolytic plants also generate blast furnace slag in preparing copper for electrolytic refining which is of the same nature as that from pyrometallurgical plants.

1.3.3 Best Technology Currently Employed (Level II)

Pyrometallurgical. Open dumping of blast furnace slag is the only disposal practice and is therefore Level II technology.

Electrolytic. The use of unlined lagoons for spent electrolyte wastewater sludge disposal and open dumping of blast furnace slag is Level II technology.

1.3.4 Technology to Provide Adequate Health and Environmental Protection (Level III)

Pyrometallurgical. Solubility tests as described in Appendix B indicated that zinc, copper, lead and other heavy metals may leach from blast furnace slag. In this event, in permeable soils, the ground would need sealing with bentonite or other sealant and collection ditches installed to collect runoff.

Electrolytic. Discarded blast furnace slag would have to be handled in the same way as slag from strictly pyrometallurgical plants (i.e. surface sealing and collection of runoff). Lined lagoons would be required to prevent percolation of leached heavy metals through permeable soils to groundwater.

Features of Levels I, II and III treatment and disposal technology are summarized in Tables 104 and 105.

1.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. The typical secondary copper pyrometallurgical processing plant has an annual production of 10,000 MT of copper alloy per year. The blast furnace is operated about 100 days per year. About 1500 MT of "black copper" is produced as an intermediate product.

The typical electrolytic plant produces 40,000 MT of 99.9% pure electrolytic (i.e. cathode) copper per year and operates 360 days.

1.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Pyrometallurgical Refining. Slag produced totals 3,500 MT/year or 35 MT/operating day. Equipment and personnel requirements are 200 hours each of truck and front loader time, and 40 hours of bulldozer time/year.

The density of the slag is 2000 kg/m^3. An area of .4 ha is needed to store 20 years of waste assuming a height of 10 m. The dump site is located on semi-industrial land on the outskirts of an urban area.

Table 104

Treatment and Disposal Technology Levels

Smelting and Refining Category Secondary Copper SIC 33412

BLAST FURNACE SLAG

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Large chunks of dense hard silicate slag; small quantities of heavy metal compounds (silicates, oxides)	Same as I	Same as I
Amount of Waste (kg/MT Product)	350 kg	350 kg	350 kg
Factors Affecting Hazardousness	Contain heavy metals Cu, Zn, Pb, N; which may leach	Same as I	Same as I
Treatment/ Disposal Technology	Open dump	Open dump	Ground sealing and collection of runoff
Estimate of # + % of Plants Using Technology	51 (100%)	51 (100%)	0 (0%)

Table 104 (cont.)

BLAST FURNACE SLAG - continued

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if heavy metals are leached	Same as Level I	Adequate
Problems and Comments	Dense silicates expected to resist leaching	Same as Level I	Same as I
Non-Land Environmental Impact	Possible groundwater or surface water contamination if leachable	Same as Level I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Monitoring wells and monitoring of runoff
Energy Requirements	Negligible	Negligible	Negligible

Table 105

Treatment and Disposal Technology Levels

Smelting and Refining Category Secondary Copper SIC 33412

Electrolytic Sludge

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Fine grained sludge with high water content (approx. 70%)	Same as I	Same as I
Amount of Waste (kg/MT Product)	0.4	0.4	0.4
Factors Affecting Hazardousness	Contains heavy metals which may leach		
Treatment/ Disposal Technology	Unlined lagoons	Unlined lagoons	Lined lagoons
Estimate of # + % of Plants Using Technology	100% (3)	100% (3)	0% (0)

Table 105 (cont'd.)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate	Inadequate	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	Possible contamination of ground or surface water	Same as I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Surveillance sump under lined lagoon
Energy Requirements	Negligible	Negligible	Negligible

There are no other disposable wastes. Costs of Level I treatment and disposal technology are given in Table 106.

Electrolytic Refining. The wastes generated consist of 56.7 MT of slag and 0.11 m^3 of sludge, daily. The slag is deposited at an on-site dump. Annual equipment and personnel requirements for disposing of the slag are 1,000 hours of front loader and truck time, and 200 hours of bulldozer time at the slag dump. The slag dump, sized to contain 20 years of waste, extends over a 1.6 ha area.

The sludge is piped to a holding pond designed to contain 20 years of waste. It characteristics are:

Volume	1,320 m^3
Bottom Width	15.2 m
Top Width	23.2 m
Bottom Length	30.4 m
Top Length	38.4 m
Total Depth	2 m
Depth of Excavation	1.1 m
Circumference	135 m
Dike Volume	581 m^3
Dike Surface	1,269 m^2
Total Width	32.8 m
Total Length	48.0 m
Required Area	0.16 ha

Level I capital and annual costs are presented in Table 107.

1.4.2 Cost of Best Technology Currently Employed (Level II)

The technology and costs of Level II are the same as Level I.

1.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection (Level III)

Pyrometallurgical Refining. The ground is sealed at the slag dump; collection ditches, pump and piping are installed. Costs incurred are:

Capital cost

Land sealing	$ 8,000
Collection ditches	960
Pump & piping	7,910
Total	$ 16,870

TABLE 106

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY SECONDARY COPPER PLANT (PYROMETALLURGICAL REFINING)

Capital Cost

	Slag
Equipment	
Truck (10%)	$ 2,500
Frontloader (10%)	2,000
Bulldozer (2%)	320
Dump	
Survey	250
Land	1,580
Total	$ 6,650

Annual Cost

Land	$ 160
Amortization	
Construction	30
Equipment	770
Operating Personnel	4,805
Repair and Maintenance	
Construction	-
Equipment	240
Energy	
Fuel	585
Electricity	60
Taxes	40
Insurance	65
Total	$6,755

TABLE 107

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY SECONDARY COPPER PLANT (ELECTROLYTIC REFINING)

Capital Cost

	Sludge	Slag
Lagoon		
Site Preparation		
Survey	$ 100	
Test Drilling	490	
Sample Testing	250	
Report Preparation	1,200	
Construction		
Excavation & Forming	775	
Compacting	1,075	
Fine Grading	570	
Soil Poisoning	165	
Transverse Drain Fields	150	
Land	635	
Equipment		
Truck (50%)		$12,500
Frontloader (50%)		10,000
Bulldozer (10%)		1,600
Dump		
Survey		1,190
Land		7,515
Total	$5,410	$32,805

Annual Cost

	Sludge	Slag
Land	$ 65	$ 750
Amortization		
Construction	555	140
Equipment	-	3,830
Operating Personnel	-	24,030
Repair and Maintenance		
Construction	80	-
Equipment	-	1,205
Energy		
Fuel	-	2,930
Electricity	-	295
Taxes	15	190
Insurance	55	330
Total	$ 770	$33,700

Annual cost

Construction Amortization	$ 1,040
Equipment Amortization	1,260
Construction Repair & Maintenance	270
Equipment Repair & Maintenance	395
Energy	
Electricity	30
Insurance	170
Total	$ 3,165

Electrolytic Refining. The ground is sealed at the slag dump; collection ditches, pump and piping are installed. The sludge is piped to a lined lagoon. Level III capital and annual costs are shown in Table 108.

Costs of Level I, II and III treatment and disposal technologies are summarized in Table 109 for secondary copper from pyrometallurgical refining and in Table 110 for secondary copper from electrolytic refining. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of copper product. Costs for each of the types of wastes (i.e. slag from pyrometallurgical refining and sludge and slag from electrolytic refining) at each level of technology are also expressed as percentages of metal selling price.

Total industry costs for Levels I, II and III technologies have been estimated in 1973 dollars and also are in Tables 109 and 110 for pyrometallurgical and electrolytic categories respectively. The industry annualized cost for Levels I and II disposal of slag from pyrometallurgical plants is estimated as $210,000 or 0.05% of estimated 1973 sales. The cost of Level III technology (i.e. adequate for environmental protection) is estimated as $300,000 or 0.08% of estimated 1973 sales value.

The industry annualized cost for Level I and II treatment and disposal of slag and sludge from electrolytic refiners is estimated as $120,000 or 0.06% of estimated national sales. The cost of Level III technology is estimated as $170,000 or 0.08% of estimated 1973 sales.

TABLE 108

COST OF LEVEL III TREATMENT AND DISPOSAL TECHNOLOGY SECONDARY COPPER PLANT (ELECTROLYTIC REFINING)

Capital Cost

	Sludge	Slag
Construction		
Lagoon Liner	$5,360	
Land Sealing		$38,000
Collection Ditches		2,090
Equipment		
Pump and Piping		11,410
Total	$5,360	$51,500

Annual Cost

	Sludge	Slag
Land	-	-
Amortization		
Construction	$ 620	$ 4,650
Equipment	-	1,815
Operating Personnel	-	-
Repair and Maintenance		
Construction	160	1,205
Equipment	-	570
Energy		
Fuel	-	-
Electricity	-	95
Taxes	-	-
Insurance	55	515
Total	$ 835	$ 8,850

TABLE 109. COST SUMMARY FOR PYROMETALLURGICAL SMELTING AND REFINING OF SECONDARY COPPER

Annual Production: Model Plant: 10,000 MT Industry: 302,000 MT

Waste (Type)	Amount
Slag	0.35

Cumulative Unit Waste Disposal Costs:

Waste (Type)	Level I $/MT of Waste	Level I $/MT of Prod.	Level II $/MT of Waste	Level II $/MT of Prod.	Level III $/MT of Waste	Level III $/MT of Prod.
Slag						
Capital Cost	$1.90	$0.67	$1.90	$0.67	$6.72	$2.35
Annual Cost	1.93	0.68	1.93	0.68	2.83	0.99
Total Capital Cost	-	0.67	-	0.67	-	2.35
Total Annual Cost	-	0.68	-	0.68	-	0.99

Cum. Industry Waste Disposal Costs ($ Million)

Waste (Type)	Level I Cap.	Level I Ann.	Level II Cap.	Level II Ann.	Level III Cap.	Level III Ann.
Slag	$0.20	$0.21	$0.20	$0.21	$0.71	$0.31
Total						

1973 Metal Price: $1,309/MT

Treatment Cost As Percent of Metal Selling Price

Waste (Type)	Level I Cap.	Level I Ann.	Level II Cap.	Level II Ann.	Level III Cap.	Level III Ann.
Slag	0.05%	0.05%	0.05%	0.05%	0.18%	0.08%
Total						

TABLE 110. COST SUMMARY FOR ELECTROLYTIC SMELTING AND REFINING OF SECONDARY COPPER

Annual Production: Model Plant: 40,000 MT Industry: 147,000 MT

Waste (Type)	Amount (MT/MT of Production)
Slag	0.43
Sludge	0.0012

Cum. Unit Waste Disposal Costs:

	Level								
Waste (Type)	I			II			III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
	Dry	Wet		Dry	Wet		Dry	Wet	
Slag Capital Cost	$1.93	-	$0.82	$1.93	-	$0.82	$4.96	-	$2.11
Slag Annual Cost	1.98	-	0.84	1.98	-	0.84	2.50	-	1.06
Sludge Capital Cost	112.71	$37.57	0.14	112.71	$37.57	0.14	224.38	$74.79	0.27
Sludge Annual Cost	16.04	5.35	0.02	16.04	5.35	0.02	33.43	11.14	0.04
Total Capital Cost	-		0.96	-		0.96	-		2.28
Total Annual Cost	-		0.86	-		0.86	-		1.10

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	$0.12	$0.12	$0.12	$0.12	$0.31	$0.16
Sludge	0.02	1/	0.02	1/	0.04	0.01
Total	0.14	0.12	0.14	0.12	0.35	0.17

1973 Metal Price: $1,309/MT

Treatment Cost as a Percent of Metal Selling Price

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	0.06%	0.06%	0.06%	0.06%	0.16%	0.08%
Sludge	0.01	1/	0.01	1/	0.02%	1/
Total	0.07	0.06	0.07	0.06	0.18	0.08

1/ negligible

2.0 SECONDARY SMELTING AND REFINING OF LEAD (SIC 33413)

2.1 INDUSTRY CHARACTERIZATION

There are approximately 82 secondary lead smelters in the United States. Production capacities of individual plants varies from about 2000 MT of lead per year to about 40,000 MT. The geographical distribution of secondary lead smelters by state and EPA region is given in Table 111. Capacities of 30 secondary lead smelters were not available. The 1972 minerals yearbook gives the 1972 production of secondary lead as approximately 553,000 MT inclusive of about 7,000 MT recovered at primary smelters (Ref. 1). The production of secondary lead has remained fairly constant at about 540,000 MT/year since 1969. The 1973 recovery of lead at secondary smelters was 593,568 MT valued at $213,166,000 (Ref. 2) which represents an estimate of national sales.

Scrap battery waste comprises by far the most important supply source for recovery of secondary lead. The recovered lead is most frequently returned to use in battery manufacture. Battery plate grids are fabricated of hard lead containing about 5% antimony. The lead oxide paste used in the plates must be manufactured from antimony-free soft lead. Both soft and hard lead can be produced from scrap battery waste. Soft lead is recovered by reverberatory smelting, and the recovered metal is converted to lead oxide either by the Barton process or by a milling process. The reverberatory slag is then smelted in blast furnace for antimonial lead recovery using scrap iron as reducing agent. Alternatively, scrap battery waste may be smelted directly in the blast furnace and only hard lead is produced.

2.2 WASTE CHARACTERIZATION

This section contains descriptions of production technology at secondary lead smelters and the resultant byproducts or wastes which are either recycled directly, shipped to other smelters for further metallic recovery or disposed of on land. Estimates are given for the quantities of wastes and potentially hazardous constituents thereof which are disposed of on land either in lagoons or open dumps.

2.2.1 Process Description. Secondary lead smelting operations differ somewhat according to the type of products produced. The three major typical operations product are the following:

Hard lead (antimonial lead) - contain approximately 5% antimony

Soft lead - pure lead

White Metal - lead-tin alloys (approximately 20% tin, 80% lead)

Table 111
GEOGRAPHIC DISTRIBUTION OF SECONDARY LEAD PRODUCTION CAPACITY

STATE	NO. OF PLANTS	PLANT DISTRIBUTION BY CAPACITY (MT)				PLANT DISTRIBUTION BY PROCESS	CAPACITY PER PROCESS (MT)
		0.025K-1K	1K-10K	10K-30K	30K-50K	P	P
ALA.	2			1		2	18,140
CALIF.	8		2	1		8	23,040
COLO.	2	2				2	230
DEL.	1					1	
FLA.	3		3			3	19,230
GA.	3			2		3	25,400
ILL.	7	2	1	2		7	30,000
IND.	4		1	2		4	46,560
KY.	1		1			1	1,630
LA.	2		1		1	2	44,450
MD.	1		1			1	1,480
MASS.	2	2				2	100
MICH.	4		2			4	15,880
MINN.	1			1		1	10,900
MISS.	1		1			1	4,540
MO.	2		1			2	
NEB.	2					2	
N.J.	3	1	1	1		3	20,960
N.Y.	4	1		1		4	18,300
N.C.	2	1				2	540
OHIO	6	2	1	1		6	19,820
PA.	7	3	1	1		7	31,890
TX.	9	1	2	2		9	35,570
TENN.	2		1	1		2	14,880
VA.	1					1	
WASH.	1					1	
WISC.	1					1	
EPA REGION							
I	2	2				2	100
II	7	2	1	2		7	39,260
III	11	3	3	1		11	35,000
IV	13	1	5	4		13	82,730
V	23	4	5	6		23	123,160
VI	11	1	3	2	1	11	78,020
VII	4		1			4	
VIII	2	2				2	230
IX	8		2	1			23,040
X	1					1	
NATIONAL	82	15	20	16	1		381,540*

P – PYROMETALLURGICAL

*CAPACITY DATA NOT AVAILABLE FOR 30 PLANTS

Based on the median value of plant size distribution the "typical" secondary lead smelter consuming scrap battery waste may be assumed to have an annual production volume of 20,000 metric tons total lead products. Any or all of the above products may be produced at individual plants. The distribution of products between soft lead, antimonial lead and lead oxide may vary widely. There are about 10 plants capable of producing both soft and antimonial lead as shown in Figure 24 and another 10 plants which produce only antimonial lead as shown in Figure 25. The "typical" plant is assumed to produce either 20,000 MT antimonial lead annually, or 10,000 MT antimonial lead and 10,000 MT soft lead and lead oxide annually. Lead recovery from discarded storage battery plates represents well over 50% of secondary lead recovered. Individual plants may have either reverbatory furnaces for soft lead production or cupola or blast furnaces for antimonial lead production or a combination thereof for both soft and antimonial lead production. Both the reverberatory and the blast furnaces are continuously operated. Assuming a normal ten percent shutdown, the daily total lead production volume would therefore amount to about 60 MT.

Secondary white metal smelting consists in the recovery of lead-tin alloys from solder, babbit, and type metal scrap waste. The metal recovered from this scrap source consists of tin-lead alloys averaging about 20% tin and 80% lead, and it is known as white metal. For high grade scrap, white metal is recovered simply by remelting in a pot furnace. For low grade drosses and skimmings, the metal is recovered by smelting in a reverberatory furnace with limestone, silica and iron scale flux. Coal fines are charged as fuel and reducing agent.

White metal is widely recycled in relatively small quantities by pot furnace melting. Reverberatory smelting of drosses and skimming from which metal recovery is performed in only a few secondary smelting plants. All of the plants in this category are of nearly the same size and the "typical" secondary white metal reverberatory smelting may be assumed to have an annual production capacity of about 9,000 metric tons. Reverberatory white metal smelting is operated continuosly. Daily production volume is about 27 MT.

2.2.2 Description of Waste Streams.

Soft and Antimonial Lead. Figure 24 illustrates processes for plants which produce both soft and antimonial lead. In making soft lead, byproduct slag is sent to a blast or cupola furnace as input for antimonial lead (i.e., hard lead) production. The emissions from the reverberatory furnace are collected in a baghouse and immediately recycled. At the present time, therefore, there is no land disposed waste except for one plant in Pennsylvania which uses a lime SO_2 scrubber system resulting in a sulfate sludge which is land disposed.

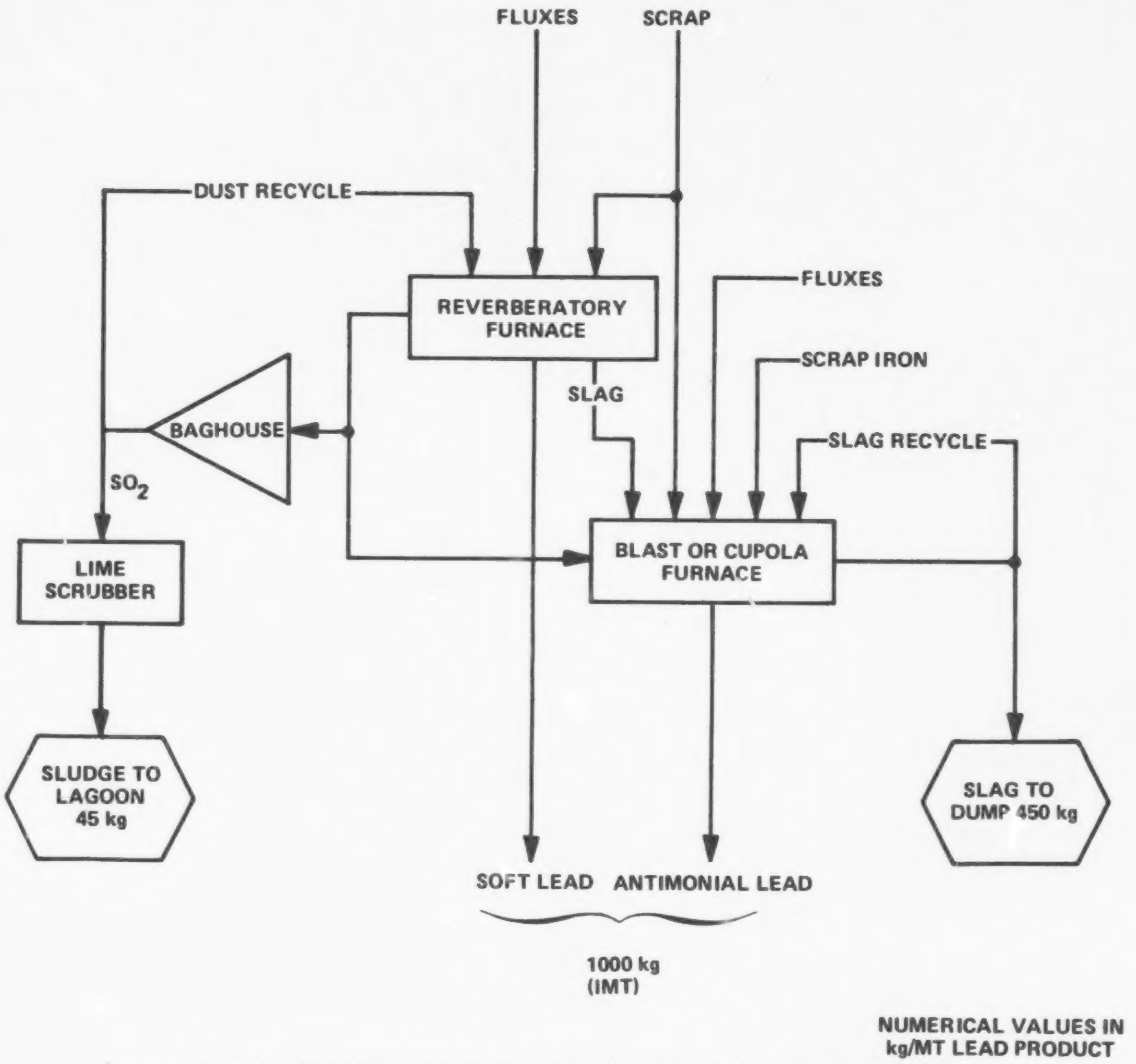

FLUXES
SCRAP
DUST RECYCLE
REVERBERATORY FURNACE
FLUXES
SCRAP IRON
SLAG
BAGHOUSE
SLAG RECYCLE
SO_2
BLAST OR CUPOLA FURNACE
LIME SCRUBBER
SLUDGE TO LAGOON 45 kg
SLAG TO DUMP 450 kg
SOFT LEAD
ANTIMONIAL LEAD
1000 kg (IMT)
NUMERICAL VALUES IN kg/MT LEAD PRODUCT

Figure 24 SECONDARY SOFT AND ANTIMONIAL LEAD SMELTING

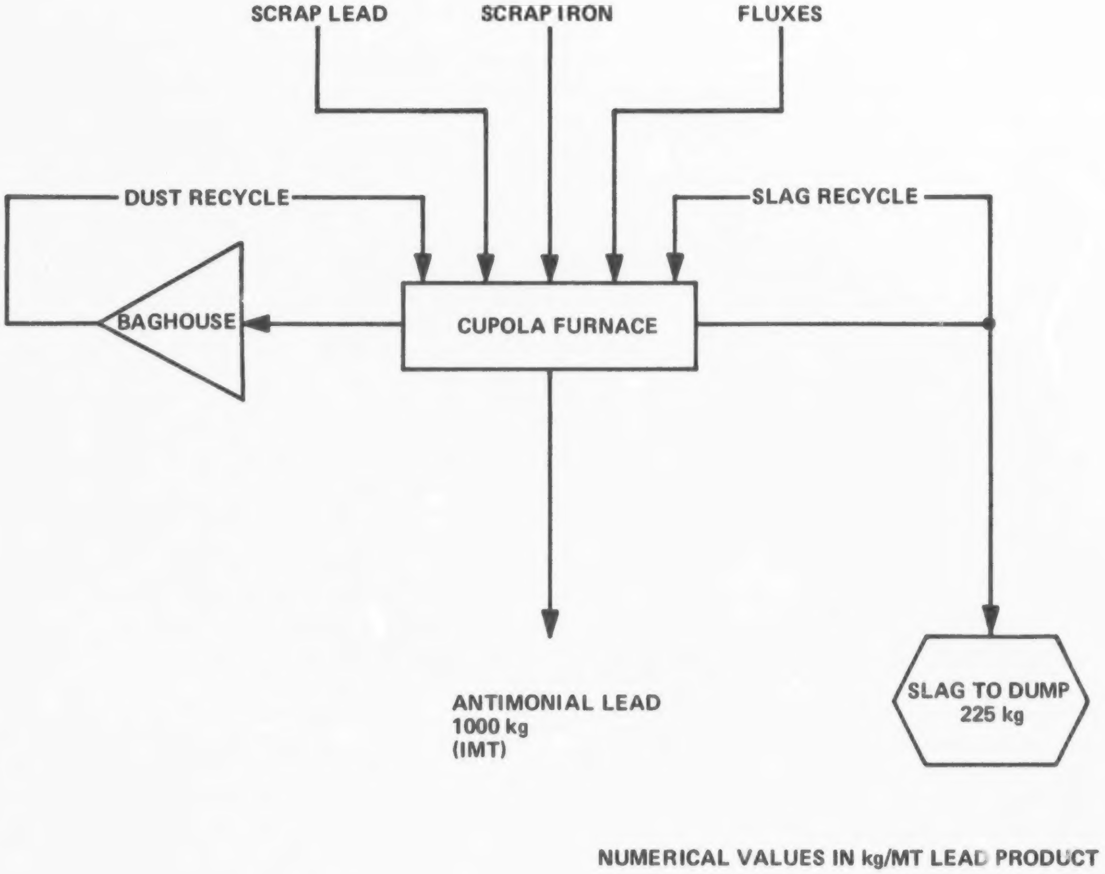

SCRAP LEAD
SCRAP IRON
FLUXES
DUST RECYCLE
SLAG RECYCLE
BAGHOUSE
CUPOLA FURNACE
ANTIMONIAL LEAD
1000 kg
(IMT)
SLAG TO DUMP
225 kg
NUMERICAL VALUES IN kg/MT LEAD PRODUCT

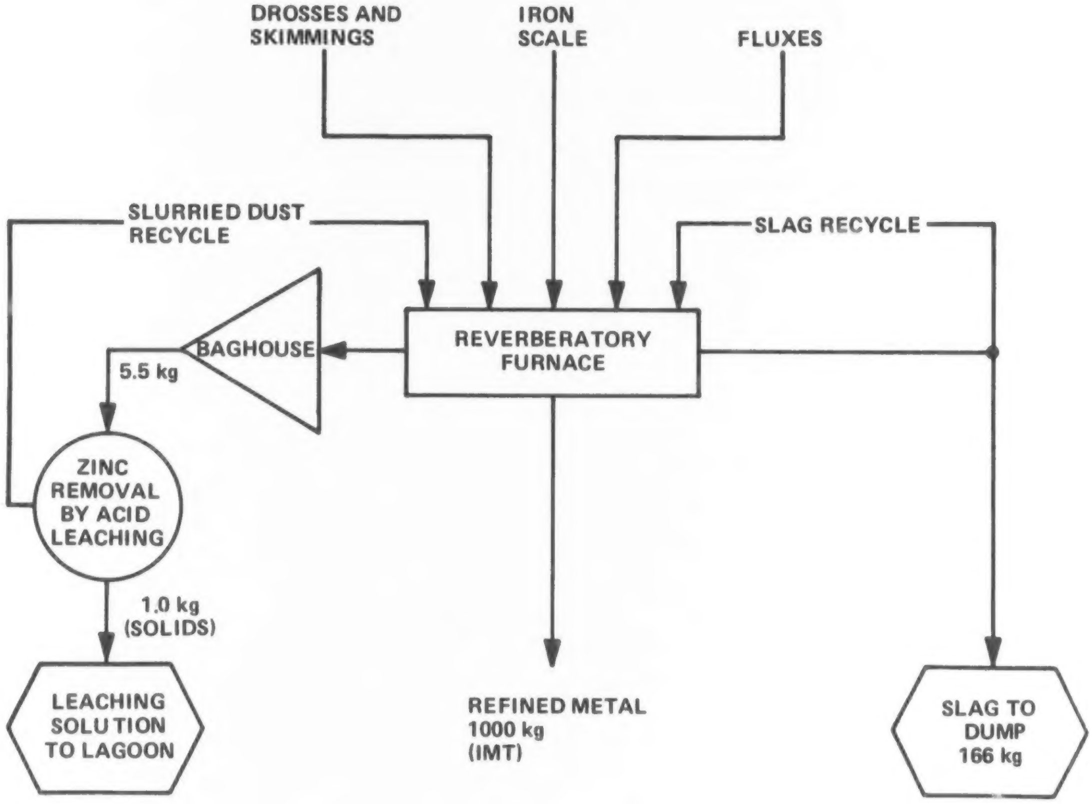
Figure 25 SECONDARY ANTIMONIAL LEAD SMELTING

SO_2 Sludge. It is expected that sulfur dioxide emissions generated from lead sulfate present in the battery plate paste and residual sulfuric acid in the scrap will be controlled by 1977 because of more stringent air pollution regulations. Lime scrubbing is most commonly used for SO_2 emission control. For the "typical" plant where 10,000 MT is produced annually, about 1,500 MT lime scrubber sludge thickened to about 30% solid content would be thus generated. This will amount to about 45 kg dry weight of lime sludge per metric ton of lead produced. Analyses of an SO_2 scrubber sludge as given in Appendix A gave high concentrations of cadmium (340 ppm), lead (53,000 ppm) and other toxic metals. In solubility tests described in Appendix B cadmium and lead were found to solubilize significantly (5 ppm Cd; 2.5 ppm Pb. For these reasons SO_2 scrubber sludge is considered potentially hazardous.

Blast Furnace Slag. In the blast furnace smelting of the resulting reverberatory slag, about 9,000 MT blast furnace slag is generated for 10,000 MT antimonial lead produced and disposed of in open dumps. This amounts to 450 kg of slag per MT of combined soft and antimonial lead production. The hard siliceous blast furnace slag is expected to have the following major chemical composition: FeO, 34.8%; CaO, 15.2%; SiO_2, 28.4%; and Pb, 1.8%. Trace metals include zinc, copper and antimony. In solubility tests described in Appendix B no metals solubilized to an extent of at least 1 ppm. This slag is therefore not considered potentially hazardous.

Antimonial Lead. With direct blast furnace smelting of scrap battery waste to produce antimonial lead (see Figure 25), the sulfur constituent in the scrap waste is scavenged by iron resulting in the formation of both a matte and a slag together with the molten metal. About 25% of this matte and slag mixture is recycled as flux for subsequent smelting. For the annual production of 20,000 MT antimonial lead, the disposable slag waste amounts to 1,800 MT annually and the matte waste, 2,700 MT annually. The quantities of disposed matte and slag are 135 and 90 kg/MT of lead product respectively. Typical chemical composition of the slag is as follows: FeO, 41.8%; SiO_2, 29.8%; CaO, 11,75%; and Pb, 0.6%. Typical chemical composition of the matte is as follows: Fe, 60.7%; Pb, 4.5%; and S, 15%. The former is typically a mixed silicate and the latter consists mainly of a mixed sulfide. The slag and the matte are tapped together and cast into a solid mass before land disposal. The slag from antimonial lead production as described previously did not solubilize trace metals to greater than 1 ppm and is not considered potentially hazardous.

White Metal. As shown in Figure 26 reverberatory furnace flue dust is controlled by baghouses. Flue dust composition will vary according to the nature of the scrap being smelted. An example of analysis is as follows: Sn, 27.7%; Sb, 0.5%; Cu, 0.06%; Pb, 17.5%; S, 6.0%; Zn, 13.3%; and Cl, 4.8%. The flue dust, because of the rather high zinc content, is not directly recyclable to the smelting furnace. It is subjected to leaching with dilute sulfuric acid for zinc removal, and the


DROSSES AND SKIMMINGS
IRON SCALE
FLUXES
SLURRIED DUST RECYCLE
SLAG RECYCLE
BAGHOUSE
REVERBERATORY FURNACE
5.5 kg
ZINC REMOVAL BY ACID LEACHING
1.0 kg (SOLIDS)
LEACHING SOLUTION TO LAGOON
REFINED METAL 1000 kg (IMT)
SLAG TO DUMP 166 kg

NUMERICAL VALUES IN kg/MT LEAD PRODUCT

Figure 26 SECONDARY WHITE METAL SMELTING

residue is recycled. The leaching solution is neutralized and stored in a lagoon. For the "typical" plant producing 9,000 MT white metal, about 50 MT fluedust is generated and treated annually. This amounts to 5.5 kg/MT lead produced.

If it is assumed that only the zinc and chloride contents of the flue dust is leached and disposed, the amount of waste disposed by lagooning (Figure 25) would amount to 9 MT for the typical plant, or 1 kg/MT product. It may be noted that currently only 4 plants are engaged in the reverberatory furnace smelting of secondary white metal. Flue dust is shipped to one plant in New Jersey for zinc leaching, and 54 MT zinc sulfate is recovered annually from the leach solution. This represents an 82% recovery of zinc value in the flue dust from these 4 plants.

The reverberatory slag is generated at the rate of about 1,500 MT per year (166 kg/MT of product) for the "typical" plant with 9,000 MT white metal annual production volume. The slag composition will vary somewhat according to the quality of scrap being smelted. As an example scrap with a high tin copper content will yield a slag with significant copper concentration. A typical set of sample analysis data is as follows: FeO, 15%; CaO, 15%; SiO_2, 25%; Al_2O_3, 5%; Sn, 2%; and Pb, 0.5%. This slag waste is tapped and cast in a large mass, and it is in a hard silicate form. On-site dumping is the most common current disposal method.

Chemical analyses of samples collected from secondary lead smelters are given in Appendix A.

2.2.3 Waste Quantities

Soft Lead and Antimonial Lead. Table 112 summarizes the generation factors for the land disposed residuals from secondary lead smelters producing soft lead and antimonial lead (i.e. blast furnace slag and SO_2 scrubber sludge). This table also gives typical concentrations of potentially hazardous constituents. Table 113 gives the yearly quantity of wastes and hazardous constituents thereof for a typical plant producing 10,000 MT of soft lead and 10,000 MT of antimonial lead per year.

Antimonial Lead. Table 112 also summarizes the generation factors for the land disposed residuals (i.e. blast furnace or cupola furnace slag) from secondary lead smelters producing only antimonial lead (i.e. hard lead). Table 113 gives the yearly quantity of slag and hazardous constituents thereof for a typical plant producing 20,000 MT of antimonial lead per year.

White Metal. Table 112 summarizes the generation factors for the land disposed residuals from secondary lead smelters producing only white metal. Table 113 gives the yearly quantity of reverbatory furnace slag from a plant producing 9,000 MT of white metal per year.

Table 112
WASTE RESIDUAL GENERATION FACTORS, SECONDARY LEAD SMELTING AND REFINING

TYPE OF WASTE	WASTE GENERATION FACTOR (Kg/MT PRODUCTION)	POTENTIALLY HAZARDOUS CONSTITUENTS CONTENT (ppm)								
		Cd	Cr	Cu	Mn	Ni	Pb	Sb	Sn	Zn
SECONDARY SOFT LEAD SMELTING:										
BLAST FURNACE SLAG	450	–	200	2,000	200	200	18,000	1,000	200	1,000
SCRUBBER SLUDGE	22 (DRY WEIGHT)	340	30	20	120	5	53,000	1,100	–	25
SECONDARY HARD LEAD SMELTING:										
CUPOLA FURNACE SLAG & MATTE	225	–	–	9,000	100	100	35,000	1,000	100	500
SECONDARY WHITE METAL SMELTING:										
REVERBERATORY FURNACE SLAG	166	5	500	120	800	5	5,000	100	20,000	500
ACID LEACH SOLIDS	1	◄—— NOT KNOWN ——►								

Table 113

ANNUAL GENERATION OF WASTE RESIDUALS FROM TYPICAL PLANTS, SECONDARY LEAD SMELTING AND REFINING*

TYPE OF WASTE	TOTAL WASTE QUANTITY (MT)	QUANTITY OF POTENTIALLY HAZARDOUS CONSTITUENTS (MT)								
		Cd	Cr	Cu	Mn	Ni	Pb	Sb	Sn	Zn
SECONDARY SOFT AND ANTIMONIAL LEAD SMELTING:										
BLAST FURNACE SLAG	9,000	–	2	18	2	2	162	10	2	10
SCRUBBER SLUDGE	1,500 (30% SOLID)	0.02	0.001	0.001	0.005	–	24	0.5	–	0.001
SECONDARY ANTIMONIAL LEAD SMELTING:										
CUPOLA FURNACE SLAG & MATTE	4,500	–	–	41	0.4	0.4	158	4	0.4	2
SECONDARY WHITE METAL SMELTING:										
REVERBERATORY FURNACE SLAG	1,500	–	0.8	0.2	1.2	–	8	0.1	30	0.8
ACID LEACH SOLIDS	9	← NOT KNOWN →								

*QUANTITIES CALCULATED FROM GENERATION AND CONCENTRATION FACTORS GIVEN IN TABLE 112 BASED ON FOLLOWING ANNUAL PRODUCTIONS:

SOFT LEAD AND ANTIMONIAL LEAD – 20,000 MT
ANTIMONIAL LEAD – 20,000 MT
WHITE LEAD – 9,000 MT
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

Using the slag and SO_2 sludge generation factors for each of the plant types previously described (i.e. soft lead and antimonial lead, antimonial lead, white metal) and the estimated production capacities of the various states for each of these types of metals estimates were made for the total quantities of land disposed slags and sludges. These estimates are given in Table 114 for 1974, 1977, and 1983.

For the purpose of hazardous waste disposal assessment, the secondary lead smelting sector may be subcategorized into soft lead, antimonial lead, and white metal smelters. Annual production volumes in each of these categories have remained essentially constant for the most recent five years from 1969 to 1973, for which statistical data are available from the U.S. Bureau of Mines (Ref. 1). Antimonial lead production, which is closely related to storage battery recycle, had exhibited a steady increase in the annual production rate prior to 1969 and again showed an appreciable gain in 1973. In view of the drop in automotive vehicle production experienced in recent years, a further growth in antimonial lead production would appear unlikely at least for the next few years. Waste projections associated with the secondary lead blast furnace slag in both 1977 and 1983 may therefore be assumed to be unchanged from current figures as presented in Table 114.

Sulfur dioxide control for secondary lead smelters was practiced at only one location in Pennsylvania in 1974. This produced 3,000 MT of lime sludge with potentially hazardous constituents. As the SO_2 air emission standards are being implemented, wet lime scrubbers would be required for all soft lead smelters with installation by 1977. This accounts for the large projected increases in sludge in 1977 as compared to 1974. Further changes by 1983 is not expected. Thus, the air emission control scrubber sludge waste projections for the secondary lead industries in both 1977 and 1983 are given in Table 114.

States with the largest quantities of total and hazardous land disposed wastes are California, Indiana, New York, Pennsylvania and Texas.

2.3 TREATMENT AND DISPOSAL TECHNOLOGY

2.3.1 Current Waste Treatment and Disposal Practices

At the present time, all discard slags from secondary lead smelting are disposed by open dumping on land.

All of the discard slags are disposed of as hard large chunks. With the exception of the antimonial lead matte which is a mixed sulfide, all discard slags are primarily mixed silicates. All slags contain significant concentrations of potentially hazardous constituents, including lead, antimony and copper. However, as previously described, leaching of these metals was not observed to a significant extent in solubility tests. Open dumping is therefore considered environmentally adequate in

Table 114-a

ESTIMATED STATE, REGIONAL AND NATIONAL WASTE FROM SECONDARY LEAD SMELTING

TOTAL BLAST FURNANCE SLAG - 1974, 1977, 1983 (METRIC TONS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	DISPOSAL METHOD	CONSTITUENTS						
				Cu	Cr	Mn	Pb	Sb	Sn	Zn
CALIFORNIA	28,500	0	OPEN DUMP	101.1	7.1	11.8	596	45.6	37.0	7.1
FLORIDA	5,400			9.7	1.4	2.2	97	9.7	1.4	1.4
GEORGIA	3,400			25.7	0.8	1.4	100	5.0	0.8	0.8
ILLINOIS	3,400			25.7	0.8	1.4	100	5.0	0.8	0.8
INDIANA	20,200			58.4	5.0	8.0	392	32.2	34.8	5.0
MINNESOTA	3,400			25.7	0.8	1.4	100	5.0	0.8	0.8
NEW JERSEY	4,900			25.9	1.2	2.0	107	5.0	31.0	1.2
NEW YORK	15,800			41.7	4.0	6.3	311	27.7	4.0	4.0
OHIO	9,900			43.7	2.3	4.0	229	16.3	2.3	2.3
PENNSYLVANIA	22,900			58.1	5.8	9.2	431	37.4	35.6	5.8
TEXAS	25,900			66.2	6.5	10.4	505	45.5	6.5	6.5
VIRGINIA	2,300			17.4	0.5	0.9	68	3.4	0.5	0.5
WASHINGTON	2,300			17.4	0.5	0.9	68	3.4	0.5	0.5
EPA REGION II	20,700			51.6	2.0	3.4	418	10.0	31.8	2.0
III	25,200			75.5	6.3	10.1	499	40.8	36.1	6.3
IV	8,800			35.4	2.2	3.6	197	14.7	2.2	2.2
V	36,900			153.5	8.9	14.8	828	58.5	38.7	8.9
VI	25,900			66.2	6.5	10.4	505	45.5	6.5	6.5
IX	28,500			101.1	7.1	11.8	596	45.6	37.0	7.1
X	2-3			17.4	0.5	0.9	68	3.4	0.5	0.5
NATIONAL TOTAL	148,300	↓	↓	520.0	37	60	3104	241	156	37

*THIS WASTE BELIEVED NONHAZARDOUS BASED ON CALSPAN SOLUBILITY TESTS DESCRIBED IN APPENDIX B.

INCLUDES SLAG FROM PLANTS PRODUCING SOFT LEAD, ANTIMONIAL LEAD AND WHITE METAL.

MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 114-b

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY LEAD SMELTERS TOTAL SCRUBBER SLUDGE, 1974 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS			
					Cd	Mn	Pb	Sb
PENN	3,000	3,000	164	LAGOON	1.0	0.3	159	3.3
EPA REGION III	3,000	3,000	164	LAGOON	1.0	0.3	159	3.3
NATIONAL TOTAL	3,000	3,000	164	LAGOON	1.0	0.3	159	3.3

*INCLUDES SLUDGE FROM ONE SMELTER WHICH PRODUCES SOFT AND ANTIMONIAL LEAD.
MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 114-c
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY LEAD SMELTING
TOTAL SCRUBBER SLUDGE - 1977, 1983 (DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS			
					Cd	Mn	Pb	Sb
CALIFORNIA	3,000	3,000	(164)	LAGOON	(1.0)	(0.3)	(159)	(3.3)
FLORIDA	900	900	(49)		(0.3)	(0.1)	(48)	(1.0)
INDIANA	2,400	2,400	(131)		(0.8)	(0.2)	(127)	(2.6)
NEW YORK	2,200	2,200	(120)		(0.7)	(0.2)	(117)	(2.4)
OHIO	900	900	(49)		(0.3)	(0.1)	(48)	(1.0)
PENNSYLVANIA	3,000	3,000	164		1.0	0.3	159	3.3
TEXAS	3,800	3,800	(207)		(1.3)	(0.4)	(201)	(4.2)
EPA REGION II	2,200	2,200	(120)		(0.7)	(0.2)	(117)	(2.4)
III	3,000	3,000	164		1.0	0.3	159	3.3
IV	900	900	(49)		(0.3)	(0.1)	(48)	(1.0)
V	3,300	3,300	(180)		(1.1)	(0.3)	(175)	(3.6)
VI	3,800	3,800	(207)		(1.3)	(0.4)	(201)	(4.2)
IX	3,000	3,000	(164)		(1.0)	(0.3)	(159)	(3.3)
NATIONAL TOTAL	16,200	16,200	(884)	↓	(5.4)	(1.6)	(859)	(17.8)

*INCLUDES ESTIMATED SLUDGE FROM PLANTS PRODUCING SOFT LEAD, ANTIMONIAL LEAD AND WHITE METAL.

MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

view of available data. Leaching solution from acid leaching of baghouse dust and scrubber sludge are currently disposed of in unlined lagoons. They are considered potentially hazardous and are discussed in further detail in the next section.

2.3.2 Present Treatment and Disposal Technology (LevelI)

Leaching Solution. At the present time leaching solution from acid leaching of baghouse dust from white metal smelting is kept in unlined lagoons (see Figure 25) and is therefore Level I practice. Solubilization of heavy metals including cadmium and lead can be expected and the use of unlined lagoons is therefore considered inadequate.

Scrubber Sludge. As discussed previously, significant quantities of lime sludge are generated if SO_2 scrubbers are installed on secondary lead smelters. Sludges are normally disposed of in unlined lagoons. This sludge contains significant concentration of cadmium and lead which can solubilize. Thus the use of unlined lagoons is not environmentally adequate.

2.3.3 Best Technology Currently Employed (Level II)

Leaching Solution. The use of unlined lagoons is the only disposal mode employed and is therefore Level II technology. This practice is environmentally inadequate as discussed previously.

Scrubber Sludge. The use of unlined lagoons is the only disposal mode employed and is therefore Level II technology. This practice is environmentally inadequate as discussed previously.

2.3.4 Technology to Provide Adequate Health and Environmental Protection (Level III)

Leaching Solution. The use of lined lagoons will prevent leaching of toxic constituents and is therefore environmentally adequate.

Scrubber Sludge. The use of lined lagoons will prevent leaching of toxic constituents and is therefore environmentally adequate.

Table 115 summarizes features of Levels I, II and III treatment and disposal technology.

2.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section only for those waste which are considered hazardous (i.e. scrubber sludge). Costs are given separately for plants which produce both soft and antimonial lead, antimonial lead only, and white metal.

Table 115

Treatment and Disposal Technology Levels

Smelting and Refining Category SECONDARY LEAD SIC 33413

SO_2 Scrubwater Sludge, Leach Sludge

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Enviromental Protection)
Physical + Chemical Properties	Aggregated soft colloidal and silt size particles; primarily calcium sulfite and sulfate. Traces of heavy metals	Same as Level I	Same as Level I
Amount of Waste (kg/MT Product)	SO_2 sludge - 45 kg Leach sludge - 1 kg	Same as I	Same as I
Factors Affecting Hazardousness	Sludge contains traces of heavy metals including Zn, Cd, Cr, Cu, Pb, Sb	Same as Level I	Same as Level I
Treatment/ Disposal Technology	Unlined Lagoon	Unlined Lagoon	Lined Lagoon
Estimate of # + % of Plants Using Technology	1 (1%)	1 (1%)	0

Table 115 (cont.)

SO_2 Scrubwater Sludge (continued)

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if heavy metals are leached	Same as I	Adequate
Problems and Comments	Only one plant scrubs SO_2 at present. All secondary lead smelters expected to scrub SO_2 by 1983.	-	-
Non-Land Environmental Impact	Possible contamination of ground and surface water if heavy metals are leached	Same as Level I	None
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Surveillance sump
Energy Requirements	Negligible	Negligible	Negligible

2.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Antimonial and Soft Lead Smelting. The typical plant producing both hard and soft lead characteristically produces 10,000 MT of soft lead and 10,000 MT of hard lead per year and operates 333 days/yr.

Scrubber Sludge. As previously discussed it is expected that the control of SO_2 emissions from secondary lead reverbatory furnaces by 1983 will result in a lime sludge. The typical plant producing 10,000 MT of hard lead and 10,000 MT of soft lead per year will generate 1500 MT of sludge with a 30 percent solid content which is hauled by tank truck to an on-site lagoon. The sludge density is about 1250 kg/m^3. Thus 1200 m^3 of sludge are disposed each year.

The lagoon is designed to hold 20 years of waste. Its design characteristics are:

Volume	30,000 m^3
Bottom width	66 m
Top width	78 m
Bottom length	132 m
Top length	144 m
Total depth	3 m
Depth of excavation	.8 m
Circumference	457 m
Dike volume	7,443 m^3
Dike surface	6,605 m^2
Total width	93 m
Total length	159 m
Required area	1.5 ha

Four test holes are drilled and 8 soil samples analyzed as part of site preparation.

One tank truck hour/day is assigned to haul the daily sludge produced, 4.1 m^3, to the lagoon.

Capital and annualized costs are given in Table 116.

Antimonial Lead. The plant has an annual capacity of 20,000 MT operating 330 days/year. Blast furnace smelting results in 1,800 MT of slag waste and 2,700 MT of matte waste annually. The resultant daily waste production is 5.45 MT and 8.1 MT, respectively. There are no costs attributable to hazardous waste disposal since the slag is not considered hazardous.

White Metal. Annual production is 9000 MT based on 333 operating days. Waste generated consists of 1500 MT of slag and 50 MT of flue dust. The dust is shipped for reclamation; waste slag is not hazardous.

Table 116

COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY SECONDARY ANTIMONIAL AND SOFT LEAD

	Capital Cost
Lagoon	
Site Preparation	
Survey	$ 940
Test Drilling	980
Sample Testing	500
Report Preparation	1,500
Construction	
Excavation & Forming	9,900
Compacting	14,325
Fine Grading	2,970
Soil Poisoning	565
Transverse Drain Fields	1,285
Land	5,935
Equipment	
Tank Truck (12.5%)	5,000
Total	$43,900

	Annual Cost
Land	$ 595
Amortization	
Construction	3,825
Equipment	795
Operating Personnel	4,045
Repair and Maintenance	
Construction	870
Equipment	250
Energy	
Fuel	535
Electricity	55
Taxes	150
Insurance	440
Total	$11,560

2.4.2 Cost of Best Technology Currently Employed (Level II)

Antimonial and Soft Lead Smelting. The technology and costs of Level II are the same as Level I for disposal of scrubber sludge.

Antimonial Lead . There are no hazardous wastes and therefore no attributable costs.

White Metal. The technology and costs of Level II are the same as Level I.

2.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection (Level III).

Antimonial and Soft Lead Smelting. This entails lining the lagoon used for scrubber sludge disposal. Associated costs are:

Capital Cost	Sludge
Lagoon liner	$ 51,420
Total	$51,420
Annual Cost	
Construction Amortization	$ 5,965
Construction Maintenance & Repair	1,545
Insurance	515
Total	$ 8,025

Costs of Level I, II and III treatment and disposal technologies are summarized in Table 117 for sludge from secondary antimonial (i.e. "hard") lead and soft lead smelting and refining. Costs are given in dollars per metric ton of lead product. Costs for each level of technology are also expressed as a percentage of metal selling price.

Total industry costs for levels I, II, and III treatment and disposal technologies have been estimated in 1973 dollars and are presented in Table 117. The annualized industry cost for Levels I and II treatment and disposal technology is estimated as $100,000 which is 0.16% of estimated 1973 sales value. The cost of Level III technology (i.e. necessary for environmental protection) is estimated as $170,000 or 0.27% of estimated 1973 sales value.

Table 117 COST SUMMARY FOR SECONDARY ANTIMONIAL AND SOFT LEAD SMELTING AND REFINING

Annual Production: Model Plant 20,000 MT Industry 169,204 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.075

Cum. Unit Waste Disposal Costs:

Waste (Type)	Level I			Level II			Level III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$29.27	$9.76	$2.20	$29.27	$9.76	$2.20	$63.55	$21.18	$4.77
Annual Cost	7.71	2.57	0.58	7.71	2.57	0.58	13.06	4.35	0.98
Total Capital Cost	-		2.20	-		2.20	-		4.77
Total Annual Cost	-		0.58	-		0.58	-		0.98

Cum. Industry Waste Disposal Costs ($ Million)

Waste (Type)	Level I		Level II		Level III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$0.37	$0.10	$0.37	$0.10	$0.81	$0.17
Total	0.37	0.10	0.37	0.10	0.81	0.17

1973 Metal Price: $358.38/MT

Treatment Cost as a Percent of Metal Selling Price

Waste (Type)	Level I		Level II		Level III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.61%	0.16%	0.61%	0.16%	1.33%	0.27%
Total	0.61	0.16	0.61	0.16	1.33	0.27

3.0 SECONDARY SMELTING AND REFINING OF ALUMINUM (SIC 33417)

3.1 INDUSTRY CHARACTERIZATION

The secondary aluminum smelting and refining industry is characterized by a large number of relatively small operations in urban areas throughout the United States. Many of these plants are engaged in only high grade scrap remelting with little attendant wastes. Plant capacities vary from 1000 MT per year to 55,000 MT per year with about 20,000 MT capacity for the typical capacity of a low grade scrap recovery operation.

Table 118 gives the estimated state, regional and national distribution of secondary aluminum smelters. Production capacities of individual plants is held proprietary by the industry. The Aluminum Recycling Association, Washington, D.C., provided information on EPA regional weekly production capacities. Based on 50 weeks per year operation, estimates of EPA regional and national production capacity are given in Table 119.

National production of aluminum from secondary smelters for 1971 and 1972 is given as 740,623 and 857,964 metric tons, respectively (Reference 1). The recovery of aluminum from secondary smelters in 1973 was 943,314 MT at an average value of 29 cents/lb (Ref. 2). Thus the value of national sales are estimated as $601,834,000.

3.2 WASTE CHARACTERIZATION

This section contains descriptions of production technology at secondary aluminum smelters and the resultant byproducts of wastes which are either recycled directly, shipped to other smelters for further metallic recovery or disposed of on land. Estimates are given for the quantities of wastes and potentially hazardous constituents thereof which are disposed of on land either in lagoons or open dumps.

3.2.1 Process Description. A generalized secondary aluminum smelting process flow diagram is shown in Figure 27.

Scrap melting. High grade aluminum scrap can be easily recycled by remelting in pot or rotary furnaces. The smelting of low grade scraps and foundry drosses is performed with reverberatory or rotary furnaces. Fluxing agents and alloying agents are charged with the scrap. Magnesium is removed by AlF_3 or Cl_2 demagging. Degassing is susually achieved simultaneously with the demagging operation. Upon completion of the smelting operation, the fluxing agent is skimmed and the purified metal melt is poured and cast into ingots. Common salt and potash mixtures are normally employed as fluxing agents.

TABLE 118

GEOGRAPHICAL DISTRIBUTION OF SECONDARY ALUMINUM SMELTERS

STATE	NO. OF PLANTS
Alabama	6
Arkansas	3
California	16
Connecticut	4
Delaware	1
Florida	1
Illinois	15
Indiana	7
Kansas	1
Kentucky	1
Maryland	2
Massachusetts	1
Michigan	3
Missouri	2
Nebraska	1
New Jersey	3
New York	9
Ohio	18

TABLE 118 (cont.)

STATE	NO. OF PLANTS
Pennsylvania	6
South Carolina	2
Tennessee	1
Texas	2
Virginia	1
Washington	2
Wisconsin	1

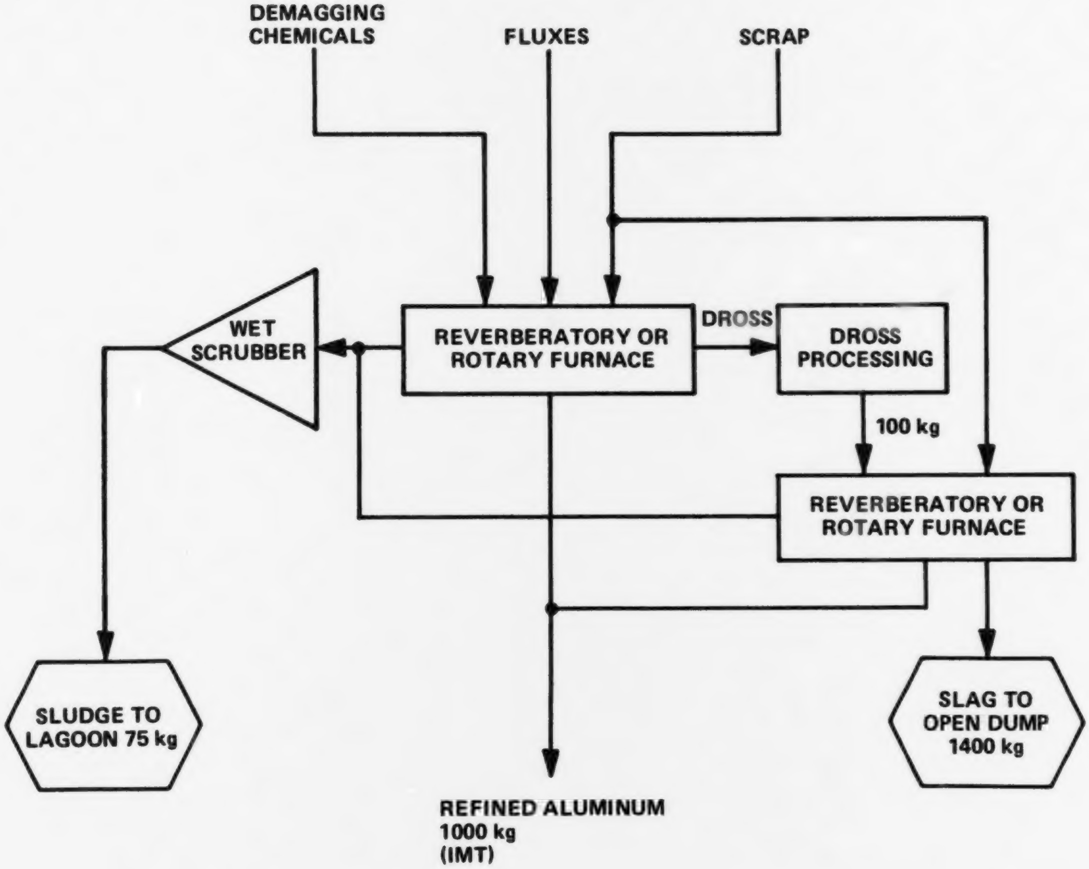
EPA REGION	NO. OF PLANTS
I	5
II	12
III	10
IV	10
V	45
VI	5
VII	4
VIII	0
IX	16
X	2
NATION	109

TABLE 119

REGIONAL AND NATIONAL PRODUCTION CAPACITY FOR SECONDARY ALUMINUM *

EPA REGION	PRODUCTION CAPACITY (METRIC TONS/YEAR)
I & II	82,220
III	29,030
IV	62,600
V	520,500
VI & VII	58,510
IX	72,580
NATIONAL	825,440

* Calculated from Data Supplied By Aluminum Recycling Association, March, 1975. Data is Combined For Some Regions to Avoid Disclosure of Proprietary Information.


DEMAGGING CHEMICALS
FLUXES
SCRAP
WET SCRUBBER
REVERBERATORY OR ROTARY FURNACE
DROSS
DROSS PROCESSING
100 kg
REVERBERATORY OR ROTARY FURNACE
SLUDGE TO LAGOON 75 kg
SLAG TO OPEN DUMP 1400 kg
REFINED ALUMINUM
1000 kg
(IMT)

NUMERICAL VALUES IN
kg/MT METAL PRODUCED

Figure 27 SECONDARY ALUMINUM SMELTING AND REFINING

Many of the secondary aluminum smelters in operation in the United States are engaged in little more than high grade scrap remelting, giving rise to little need for smelting waste disposal. For the purpose of assessing solid waste disposal in aluminum scrap smelting, the "typical" plant may be assumed to engage in low grade scrap recycle with fluxing and demagging capabilities. Based on the median value of plant size distribution the plant production volume is assumed to be about 20,000 MT per year or about 60 MT per day.

Dross Smelting. Secondary aluminum smelting from low grade drosses containing 10 to 30% aluminum metal values requires pre-processing of the dross to enrich the aluminum metal content to about 75%. Either wet or dry processes may be employed. In the dry process, the dross material is crushed and comminuted in impaction or ball mills. The fines are removed, leaving enriched metal granules for smelting in reverberatory or rotary furnaces. Alternatively, the dross raw material may be processed by water washing in rotating drum, where the water-soluble fluxing salts are removed. Wet processing is more commonly applied in relatively low volume dross processing plants.

The typical aluminum dross smelting plant is assumed to produce 10,000 MT aluminum metal annually from raw dross material. This is derived from typical productions of plants engaged in aluminum dross smelting. In practice dross smelting is performed in relatively large plants where the total metal production amounts to about 50,000 MT per year. For waste disposal assessment, dross smelting may be considered separately as taking place in a hypothetical plant. The daily production volume is taken as 30 MT.

3.2.2 Description of Waste Streams

Scrap melting. In either reverberatory or rotary furnace aluminum scrap smelting, the byproduct slag or dross waste will consist mainly of a salt mixture of sodium chloride (NaCl), potassium chloride (KCl) and magnesium chloride ($MgCl_2$) with about 15% aluminum value. This slag is further processed in plants with dross smelting capabilities. For the typical plant with 20,000 MT annual production volume, about 2,000 MT of the dross is generated annually (100 kg/MT of aluminum metal). This byproduct dross is currently recycled. It may present a future disposal problem if dross smelting for further metal recovery becomes non-profitable or impractical. Byproduct dross is stored in covered areas so there is no danger of salt leaching.

Another smelting activity related waste arises from air pollution control residues. Air emission control is provided mainly by wet-scrubbing with lime treatment. For the typical plant with 20,000 MT aluminum alloy production, it is estimated that about 5,000 MT sludge thickened to 30% solid content is generated annually. In solubility tests described in Appendix B fluoride was found to leach from emission control dust in significant concentration. Similar leaching is expected from emission control sludge. For this reason emission control sludge (or dusts) are considered potentially hazardous.

Dross Smelting. About 10-15% of total secondary aluminum metal is produced from dross smelting. The initial raw dross is enriched from 25% metal content to 75% metal content. In wet processing, the flux salt remaining from dross enrichment is removed by dissolution in water. In dry processing, the flux salt is recovered as Al_2O_3 which is recycled as steel melting flux cover agent. The major source of solid waste arises from both the impure constituents in the enriched dross and the salt flux used in the smelting process. The waste generated is in the form of a high salt slag consisting mainly of NaCl and KCl with about 6-8% residual aluminum value. For the typical plant production of 10,000 MT aluminum from drosses, about 14,000 MT high salt slag is generated (1400 kg/MT aluminum). Relatively small quantities of flue dust are also generated as furnace emissions, and these are combined with the slag waste. This waste is considered potentially hazardous because of the high soluble salt content (NaCl, KCl) which can easily be leached and contaminate ground water.

3.2.3 Waste Quantities. Table 120 summarizes the generation factors for the land disposed residuals (i.e. scrubber sludge, high salt slag) from secondary aluminum smelting operations. This table also gives typical concentrations of potentially hazardous constituents. Table 121 gives the yearly quantity of wastes and hazardous constituents thereof for a typical plant producing 20,000 MT of aluminum alloy metal from scrap metal and 10,000 MT of aluminum metal from aluminum drosses. Appendix A gives analyses of samples collected from secondary aluminum smelters.

Using the sludge and high salt slag generation factors given in Table 120 and concentration factors given in Table 120 along with the capacity of each state for secondary aluminum production estimates have been made for total and hazardous waste generation. These estimates are given for 1974, 1977, and 1983 in Table 122.

Projections of secondary aluminum production volumes for the years 1977 and 1983 cannot be derived by simple extrapolation of past production data. The annual consumption of aluminum has been increasing exponentially at the rate of about 6% per year. Primary aluminum production capacity has kept pace more as a step function. Thus, the secondary smelter production, in satisfying the total market demand, had to contend with occasional primary smelting overcapacities and metal stockpiles. As a result, secondary aluminum annual production volume growth has followed a rather regular step function pattern of making a 20% increase after about every three years at a steady production rate. It appears that, even though the growth in aluminum consumption and primary smelting will level off somewhat, secondary aluminum smelting will continue to grow at the exponential rate of 7% per year. The projected 1977 secondary aluminum production volume would be about 20% higher than that in 1974, and the 1983 volume would be about 70% higher. Both the smelter process technology and air emission control technology are not expected to change significantly within the short term. Therefore, both the air

TABLE 120 WASTE RESIDUAL GENERATION FACTORS, SECONDARY ALUMINUM METAL SMELTING

Type of Waste	Waste Generation Factor (Kg/MT Production	Potentially Hazardous Constituents Content (ppm)					
		Cr	Cu	Mn	Ni	Pb	Zn
Secondary Aluminum Scrap Smelting							
Scrubber Sludge	75	20	1,250	-	-	140	6,500
Secondary Aluminum Dross Smelting							
High Salt Slag	1,400	60	310	100	10	300	240

TABLE 121 YEARLY GENERATION OF WASTE REDISUALS FROM TYPICAL PLANT,* SECONDARY ALUMINUM METAL SMELTING (SIC 3341)

Type of Waste	Total Waste Disposed (MT)	Quantity of Potentially Hazardous Constituents (MT)					
		Cr	Cu	Mn	Ni	Pb	Zn
Secondary Aluminum Scrap Smelting							
Scrubber Sludge	1,500	0.03	1.9	-	-	0.2	9.8
Secondary Aluminum Dross Smelting							
High Salt Slag	14,000	0.8	4.3	1.4	0.1	4.2	3.4

* Quantities calculated from generation and concentration factors in Table 120 based on annual production of 20,000 MT aluminum scrap and 10,000 MT aluminum from dross. Mulitply sludge values by 3.0 to obtain approximate wet weights. Divide by 250 to obtain daily quantities. Multiply by 1.1 to convert to short tons.

Table 122-a

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY ALUMINUM SMELTING TOTAL SCRUBBER SLUDGE, 1974 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS		
					Cu	Pb	Zn
ALABAMA	3,100	3,100	24.4	LAGOON	3.9	0.4	20.1
ARKANSAS	2,500	2,500	19.6		3.1	0.3	16.2
CALIFORNIA	10,000	10,000	78.9		12.5	1.4	65.0
CONNECTICUT	2,500	2,500	19.6		3.1	0.3	16.2
ILLINOIS	12,500	12,500	98.3		15.6	1.7	81.0
INDIANA	12,500	12,500	98.3		15.6	1.7	81.0
KANSAS	1,200	1,200	9.5		1.5	0.2	7.8
MARYLAND	1,200	1,200	9.5		1.5	0.2	7.8
MICHIGAN	4,400	4,400	35.8		6.6	0.6	28.6
NEW JERSEY	2,500	2,500	19.6		3.1	0.3	16.2
NEW YORK	6,200	6,200	49.0		7.8	0.9	40.3
OHIO	31,300	31,300	246.5		39.1	4.4	203.0
PENNSYLVANIA	2,500	2,500	19.6		3.1	0.3	16.2
S. CAROLINA	2,500	2,500	19.6		3.1	0.3	16.2
TEXAS	2,500	2,500	19.6		3.1	0.3	16.2
WASHINGTON	1,200	1,200	9.5		1.5	0.2	7.8
WISCONSIN	2,500	2,500	19.6		3.1	0.3	16.2
EPA REG. I	2,500	2,500	19.6		3.1	0.3	16.2
II	8,700	8,700	68.6		10.9	1.2	56.5
III	3,700	3,700	29.1		4.6	0.5	24.0
IV	5,600	5,600	44.0		7.0	0.7	36.3
V	63,200	63,200	498.5		80.0	8.7	409.8
VI	5,000	5,000	39.2		6.2	0.6	32.4
VII	1,200	1,200	9.5		1.5	0.2	7.8
IX	10,000	10,000	78.9		12.5	1.4	65.0
X	1,200	1,200	9.5		1.5	0.2	7.8
NATIONAL TOTAL	101,100	101,100	796.9		127.3	13.8	655.8

*MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 122-b

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY ALUMINUM SMELTING TOTAL SCRUBBER SLUDGE, 1977 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS		
					Cu	Pb	Zn
ALABAMA	3,720	3,720	29.3	LAGOON	4.7	0.5	24.1
ARKANSAS	3,000	3,000	23.5		3.7	0.4	19.4
CALIFORNIA	12,000	12,000	94.7		15.0	1.7	78.0
CONNECTICUT	3,000	3,000	23.5		3.7	0.4	19.4
ILLINOIS	15,000	15,000	118.0		18.7	2.0	97.2
INDIANA	15,000	15,000	118.0		18.7	2.0	97.2
KANSAS	1,440	1,440	11.4		1.8	0.2	9.4
MARYLAND	1,440	1,440	11.4		1.8	0.2	9.4
MICHIGAN	5,280	5,280	43.0		7.9	0.7	34.3
NEW JERSEY	3,000	3,000	23.5		3.7	0.4	19.4
NEW YORK	7,440	7,440	58.8		9.4	1.1	48.4
OHIO	37,560	37,560	295.8		46.9	5.3	243.6
PENNSYLVANIA	3,000	3,000	23.5		3.7	0.4	19.4
S. CAROLINA	3,000	3,000	23.5		3.7	0.4	19.4
TEXAS	3,000	3,000	23.5		3.7	0.4	13.4
WASHINGTON	1,440	1,440	11.4		1.8	0.2	9.4
WISCONSIN	3,000	3,000	23.5		3.7	0.4	19.4
EPA REG. I	3,000	3,000	23.5		3.7	0.4	19.4
II	10,440	10,440	82.3		13.1	1.5	67.8
III	4,440	4,440	34.9		5.5	0.6	28.8
IV	6,720	6,720	52.8		8.4	0.9	43.5
V	75,840	75,840	598.3		95.9	10.4	491.7
VI	6,000	6,000	47.0		7.4	0.8	38.8
VII	1,440	1,440	11.4		1.8	0.2	9.4
IX	12,000	12,000	94.7		15.0	1.7	78.0
X	1,440	1,440	11.4		1.8	0.2	9.4
NATIONAL TOTAL	121,320	121,320	956.3		152.6	16.7	786.8

*MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 122-c

ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY ALUMINUM SMELTING TOTAL SCRUBBER SLUDGE, 1983 (METRIC TONS, DRY WEIGHTS)*

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS		
					Cu	Pb	Zn
ALABAMA	5,270	5,270	41.5	LAGOON	6.6	0.7	34.2
ARKANSAS	4,250	4,250	33.3		5.3	0.5	27.5
CALIFORNIA	17,000	17,000	134.1		21.3	2.4	110.5
CONNECTICUT	4,250	4,250	33.3		5.3	0.5	27.5
ILLINOIS	21,250	21.250	167.1		26.5	2.9	137.7
INDIANA	21,250	21,250	167.1		26.5	2.9	137.7
KANSAS	2,040	2,040	16.2		1.6	0.3	13.3
MARYLAND	2,040	2,040	16.2		2.6	0.3	13.3
MICHIGAN	7,480	7,480	60.9		11.2	1.0	48.6
NEW JERSEY	4,250	4,250	33.3		5.3	0.5	27.5
NEW YORK	10,540	10,540	83.3		13.3	1.5	68.5
OHIO	53,210	53,210	419.1		66.5	7.5	345.1
PENNSYLVANIA	4,250	4,250	33.3		5.3	0.5	27.5
S. CAROLINA	4,250	4,250	33.3		5.3	0.5	27.5
TEXAS	4,250	4,250	33.3		5.3	0.5	27.5
WASHINGTON	2,040	2,040	16.2		2.6	0.3	13.3
WISCONSIN	4,250	4,250	33.3		5.3	0.5	27.5
EPA REG. I	4,250	4,250	33.3		5.3	0.5	27.5
II	14,790	14,790	116.6		18.6	2.0	96.0
III	6,290	6,290	49.5		7.9	0.8	40.8
IV	9,520	9,520	74.8		11.9	1.2	61.7
V	107,440	107,440	847.5		136.0	14.8	696.6
VI	8,500	8,500	66.6		10.6	1.0	55.0
VII	2,040	2,040	16.2		2.6	0.3	13.3
IX	17,000	17,000	134.1		21.3	2.4	110.5
X	2,040	2,040	16.2		2.6	0.3	13.3
NATIONAL TOTAL	171,870	171,870	1,354.8	↓	216.8	23.3	1,114.7

*MULTIPLY BY 3.0 TO CONVERT TO APPROXIMATE WET WEIGHTS.
MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS.

SOURCE: CALSPAN CORPORATION

Table 122-d
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY ALUMINUM SMELTERS
TOTAL HIGH SALT SLAG, 1974 (METRIC TONS)**

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS* CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS				
					Cu	Cr	Mn	Pb	Zn
CALIFORNIA	28,000	28,000	27.7	LANDFILL	8.1	1.7	2.8	8.4	6.7
INDIANA	91,000	91,000	91.0		27.3	5.5	9.1	27.3	21.8
OHIO	112,000	112,000	112.0		33.6	6.7	11.2	33.6	26.9
PENNSYLVANIA	14,000	14,000	14.0		4.2	0.8	1.4	4.2	3.4
EPA REGION III	14,000	14,000	14.0		4.2	0.8	1.4	4.2	3.4
EPA REGION V	203,000	203,000	203.0		60.9	12.2	20.3	60.9	48.7
EPA REGION IX	28,000	28,000	27.7		8.1	1.7	2.8	8.4	6.7
NATIONAL TOTAL	245,000	245,000	244.7	↓	73.5	14.7	24.5	73.5	58.8

*BALANCE OF SLAG RESIDUE CONSISTS MAINLY OF NaCl AND KCl SALTS.

**MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

Table 122-e
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY ALUMINUM SMELTERS
TOTAL HIGH-SALT SLAG, 1977 (METRIC TONS)**

STATE		TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS* CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS				
						Cu	Cr	Mn	Pb	Zn
CALIFORNIA		34,000	34,000	33.2	LANDFILL	9.7	2.0	3.4	10.1	8.0
INDIANA		109,000	109,000	109.2	↓	32.8	6.6	10.9	32.8	26.2
OHIO		134,000	134,000	134.4		40.3	8.0	13.4	40.3	32.3
PENNSYLVANIA		17,000	17,000	16.8		5.0	1.0	1.7	5.0	4.1
EPA REGION	III	17,000	17,000	16.8		5.0	1.0	1.7	5.0	4.1
	V	243,000	243,000	243.6		73.1	14.6	24.3	73.1	58.5
	IX	34,000	34,000	33.2		9.7	2.0	3.4	10.1	8.0
NATIONAL TOTAL		294,000	294,000	293.6	↓	87.8	17.6	29.4	88.2	20.6

*BALANCE OF SLAG RESIDUE CONSISTS MAINLY OF NaCl AND KCl SALTS.

**MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

Table 122-f
ESTIMATED STATE, REGIONAL, AND NATIONAL WASTE FROM SECONDARY ALUMINUM SMELTERS
TOTAL HIGH SALT SLAG, 1983 (METRIC TONS)**

STATE	TOTAL DISPOSED	TOTAL POTENTIALLY HAZARDOUS	TOTAL HAZARDOUS* CONSTITUENTS	DISPOSAL METHOD	HAZARDOUS CONSTITUENTS				
					Cu	Cr	Mn	Pb	Zn
CALIFORNIA	48,000	48,000	47	LANDFILL	13.8	2.9	4.8	14.3	11.4
INDIANA	155,000	155,000	155		46.4	9.4	15.5	46.4	37.1
OHIO	190,000	190,000	190		57.1	11.4	19.0	57.1	45.7
PENNSYLVANIA	24,000	24,000	24		7.1	1.4	2.4	7.1	5.8
EPA REGION III	24,000	24,000	24		7.1	1.4	2.4	7.1	5.8
V	345,000	345,000	345		103.5	20.8	34.5	103.5	82.8
IX	48,000	48,000	47		13.8	2.9	4.8	14.3	11.4
NATIONAL TOTAL	417,000	417,000	416		124.4	25.1	41.7	124.9	100.0

*BALANCE OF SLAG RESIDUE CONSISTS MAINLY OF NaCl AND KCl SALTS.

**MULTIPLY BY 1.1 TO CONVERT TO SHORT TONS

SOURCE: CALSPAN CORPORATION

emission control scrubber sludge figures for scrap smelting and the high salt slag data for dross smelting for 1974 were multiplied by a factor of 1.2 to derive the corresponding 1977 projections and by 1.7 for 1983 projections.

The state of Ohio has the greatest quantity of potentially hazardous sludge generated followed by California, Illinois and Indiana. Ohio also generates the greatest quantity of potentially hazardous high salt slag followed by Indiana, California and Pennsylvania. High salt slag is generated in only four states whereas sludge is generated in 17 states. This is because only 4 states have dross processing facilities which generate high salt slag.

3.3 TREATMENT AND DISPOSAL TECHNOLOGY

3.3.1 Current Waste Treatment and Disposal Practices

Currently wet scrubber sludges from secondary aluminum smelters is usually sent to unlined lagoons. The high salt slag remaining from dross smelting is currently open dumped either on site or off site. It is estimated that on a national basis about 50% of the slag is disposed of on site and 50% off site by contract disposal service. Both of these wastes are considered potentially hazardous as discussed previously. Levels of technology for their treatment and disposal are discussed in the following sections.

3.3.2 Present Treatment and Disposal Technology (Level I)

Scrubber Sludge. At the present time scrubber sludge is usually disposed of in unlined lagoons. This practice is environmentally inadequate if fluorides leach through permeable soils to groundwater.

High salt slag. At the present time high salt slag residue from dross processing is disposed of in open dumps.

Potassium, sodium and chloride which are present at high concentrations in high salt slag are relatively non-toxic when compared to heavy metals. However, the high concentration of these constituents and their high solubility presents a potential hazard to ground water quality. Open dumping of high salt slag in permeable soil areas is therefore environmentally unacceptable.

3.3.3 Best Technology Currently Employed (Level II)

Scrubber Sludge. At least one secondary smelter uses lined lagoons for scrubber sludge disposal. The use of lined lagoons to preclude leaching of fluorides and other constituents is environmentally acceptable and comprises Level II technology.

High Salt Slag. Level II technology is the same as Level I (i.e. open dumping) and is not environmentally sound.

3.3.4 Technology to Provide Adequate Health and Environmental Protection (Level III)

Scrubber Sludge. Level III technology is the use of lined lagoons which is environmentally adequate.

High Salt Slag. Level III technology is ground sealing of high salt slag disposal areas with bentonite or other soil sealants to prevent movement of soluble salts through permeable soils.

Tables 123 and 124 summarize features of Levels I, II and III treatment and disposal technology including extent of use, factors affecting degree of hazard, and environmental adequacy.

3.4 COST ANALYSES

In the last section various treatment and disposal technologies currently employed or considered for adequate health and environmental protection were described. The costs of implementing this technology for typical plants is considered in this section. Costs are given separately for reverbatory smelter producing 20,000 MT of aluminum metal per year and the dross smelting facility producing 10,000 MT of aluminum metal.

3.4.1 Cost of Present Treatment and Disposal Technology (Level I)

Reverbatory Smelting. About 5,000 MT of scrubber water sludge with a 30 percent solid content is generated annually which is disposed in an on-site lagoon.

The estimated sludge density is 1250 kg/m^3. About 4,000 m^3 of sludge are deposited annually in the lagoon. The lagoon is designed to retain 80,000 m^3 of sludge which represents 20 years of operation. The lagoon characteristics necessary to accommodate this volume of sludge are:

Volume	100,000 m^3
Bottom width	95 m
Top width	115 m
Bottom length	189 m
Top length	209 m
Total depth	5 m
Depth of excavation	1.35 m
Circumference	659 m
Dike volume	24,772 m^3
Dike surface	14,524 m^2
Total width	135 m
Total length	230 m
Required area	3.1 ha

Table 123

Treatment and Disposal Technology Levels

Smelting and Refining Category SECONDARY ALUMINUM SIC 33417

SCRUBBER SLUDGE

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Environmental Protection)
Physical + Chemical Properties	Colloidal and silt size aggregated particles; metal oxides and hydroxides, lime	Same as I if wet scrubbing system; metal oxide dust particulates if dry dust collection	Same as II
Amount of Waste (kg/MT Product)	75 kg	75 kg for wet system greatly reduced quantity of solids as dust if dry emissions collection	Same as II
Factors Affecting Hazardousness	Contains heavy metals including Pb, Zn, Cu, Cr	Same as I	
Treatment/ Disposal Technology	Unlined lagoon	Lined lagoons or coated baghouse for dust collection	Lined Lagoons for wet scrubber sludge
Estimate of # + % of Plants Using Technology	Unknown but predominant practice is wet scrubbing	Unknown - at least one plant uses lined lagoon and at least one plant uses dry emissions control	Same as II

Table 123 (cont'd.)

SCRUBBER SLUDGE - continued

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if heavy metals leach from sludge	Adequate for lined lagoons; may need secure dust disposal if dry system used	Adequate
Problems and Comments	-	-	-
Non-Land Environmental Impact	Possible contamination of ground and surface water	None for lined lagoon; possible ground or surface water contamination from dust disposal	Adequate
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Groundwater survelliance
Energy Requirements	Negligible	Negligible	Negligible

Table 124

Treatment and Disposal Technology Levels

Smelting and Refining Category SECONDARY ALUMINUM SIC 33417

High Salt Slag

Factor	Level I (Prevalent)	Level II (Best Available)	Level III (Adequate Health and Environmental Protection)
Physical + Chemical Properties	Chunks of friable slag; predominantly NaCl, KCl with 6-8% aluminum and traces of other metals	Same as I	Same as I
Amount of Waste (kg/MT Product)	1400 kg	1400 kg	1400 kg
Factors Affecting Hazardousness	Contains heavy metals including Cu, Cr, Pb, Zn	Same as I	Same as I
Treatment/ Disposal Technology	Open dumping at off-site landfills	Same as I	Ground sealing of disposal area
Estimate of # + % of Plants Using Technology	Not known but predominant practice	Same as I	0

Table 124 (cont.)

High Salt Slag - continued

Factor	Level I	Level II	Level III
Adequacy of Technology	Inadequate if leaching of heavy metals	Same as I	Adequate for heavy metals but not for salt
Problems and Comments	NaCl and KCl not considered hazardous but have high potential for groundwater contamination due to high content in slag and high solubility.	-	No known method for sealing ground against NaCl and KCl leaching
Non-Land Environmental Impact	Possible contamination of ground and surface water	Same as I	Heavy metal leaching prevented but salt leaching not
Compatibility With Existing Facilities	Compatible	Compatible	Compatible
Monitoring & Surveillance Methods	None	None	Groundwater monitoring wells
Energy Requirements	Negligible	Negligible	Negligible

Eight test holes each 10 m deep, are drilled and 2 samples are analyzed per hole during site preparation. The lagoon is situated on semi-industrial land. Costs of Level I treatment and disposal technology are given in Table 125.

Dross Smelting. Annual production of 10,000 MT results in 14,000 MT of slag. Current practice is disposal by contractors at off-site landfills.

The slag density is 2000 kg/m^3. Contractor costs consist of loading, hauling and grading at the dump site and amount to $\$3.57/m^3$. Costs are summarized below for Level I disposal technology:

Capital Cost	N.A. 1/
Annual Cost	
Contractor	$24,990
Total	$24,990

1/ Contractor Disposal

3.4.2 Cost of Best Technology Currently Employed (Level II)

Reverbatory Smelting. At least one secondary aluminum smelter uses a lined lagoon for containment of scrubber water sludge. This practice will preclude any leaching. The added capital cost of the lagoon lining for the hypothetical plant is estimated as $106,525. Annual costs are summarized below:

Annual Cost	
Construction Amortization	$12,370
Construction Repair & Maintenance	3,195
Insurance	1,065
Total	$16,630

Dross Smelting. Disposal of high salt slag in landfills is both Level I and II practice. Thus there are no added costs for Level II.

3.4.3 Cost of Technology to Provide Adequate Health and Environmental Protection.

Reverbatory Smelting. The use of lined lagoons for scrubber water sludge is also Level III practice and there are no added costs.

Dross Smelting . The high salt slag wastes are disposed by an outside contractor. It is assumed that the contractor must secure his land-

TABLE 125 COST OF LEVEL I TREATMENT AND DISPOSAL TECHNOLOGY SECONDARY ALUMINUM PLANT - REVERBATORY SMELTING

CAPITAL COST

	Sludge
Lagoon	
Site Preparation	
Survey	$ 1,940
Test Drilling	1,960
Sample Testing	1,000
Report Preparation	1,800
Construction	
Excavation & Forming	32,945
Compacting	45,830
Fine Grading	6,535
Soil Poisoning	815
Transverse Drain Fields	3,105
Land	12,260
Total	$108,190

ANNUAL COST

Land	$ 1,225
Amortization	
Construction	11,140
Equipment	-
Operating Personnel	-
Repair and Maintenance	
Construction	2,675
Equipment	-
Energy	
Fuel	-
Electricity	30
Taxes	305
Insurance	1,080
Total	$16,455

fill in the same manner as required for on-site dumping; namely seal the ground,, construct collection ditches and install a pump and piping. The estimated cost to the plant is $10,670 per year.

Annual Cost

Construction Amortization	$ 3,460
Equipment Amortization	1,740
Construction Repair & Maintenance	895
Equipment Repair & Maintenance	545
Energy	
Electricity	80
Insurance	405
Adm. & Profit	3,565
Total	$10,670

Costs of Levels I, II and III treatment and disposal technologies are summarized in Table 126 for reverbatory smelting and Table 127 for dross smelting. Costs are given in dollars per metric ton of dry and wet waste and dollars per metric ton of aluminum product. Costs for each type of waste (i.e. sludge, high salt slag) at each technology level are also expressed as percentages of metal selling prices.

Total industry costs for Levels I, II and III treatment and disposal technologies have been estimated in 1973 dollars and are also presented in Tables 126 and 127 for reverbatory smelting and dross smelting respectively. The industry annualized cost for Level I sludge disposal from reverbatory smelting is estimated as $620,000 which represents 0.13% of 1973 estimated sales. The industry costs for Levels II and III technology are estimated as $1,250,000 and consists of lining sludge disposal lagoons for adequate environmental protection.

Total annualized industry costs for Level I and II treatment and disposal of high salt slag from dross smelting is estimated as $470,000 which represents 0.39% of estimated 1973 national sales. The annualized industry cost for Level III treatment and disposal technology (i.e. adequate for environmental protection) is $670,000 or 0.56% of estimated 1973 sales.

TABLE 126 COST SUMMARY FOR SECONDARY ALUMINUM REVERBATORY SMELTING

Annual Production: Model Plant-20,000 MT Industry-754,909 MT

Waste (Type)	Amount (MT/MT of Production)
Sludge	0.25

Cum. Unit Waste Disposal Costs:

Waste (Type)	I			II			III		
	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.	$/MT of Waste		$/MT of Prod.
Sludge	Dry	Wet		Dry	Wet		Dry	Wet	
Capital Cost	$21.64	$7.21	$5.41	$42.94	$14.31	$10.74	$42.94	$14.31	$10.74
Annual Cost	3.29	1.10	0.82	6.62	2.21	1.65	6.62	2.21	1.65
Total Capital Cost	-		$5.41	-		$10.74	-		$10.74
Total Annual Cost	-		0.82	-		1.65	-		1.65

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	$4.08	$0.62	$8.11	$1.25	$8.11	$1.25
Total	$4.08	$0.62	$8.11	$1.25	$8.11	$1.25

1973 Metal Price: $638.00/MT

Treatment cost as Percent of Metal Selling Price

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Sludge	0.85%	0.13%	1.68%	0.26%	1.68%	0.26%
Total	0.85%	0.13%	1.68%	0.26%	1.68%	0.26%

TABLE 127 COST SUMMARY FOR SECONDARY ALUMINUM DROSS SMELTING

Annual Production: Model Plant-10,000 MT Industry-188,727 MT

Waste (Type)	Amount (MT/MT of Production)
Slag	0.7

Cumulative Unit Waste Disposal Costs:

	Level					
	I		II		III	
Waste (Type)	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.	$/MT of Waste	$/MT of Prod.
Slag						
Capital Cost	-	-	-	-	-	-
Annual Cost	$3.57	$2.50	$3.57	$2.50	$5.09	$3.57
Total Capital Cost	-	-	-	-	-	-
Total Annual Cost	-	$2.50	-	$2.50	-	$3.57

Cum. Industry Waste Disposal Costs ($ Million)

	Level					
	I		II		III	
Waste (Type)	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	-	$0.47	-	$0.47	-	$0.67
Total	-	$0.47	-	$0.47	-	$0.67

1973 Metal Price: $638.00/MT

Treatment Cost As Percent of Metal Selling Price

	Level					
Waste (Type)	I		II		III	
	Cap.	Ann.	Cap.	Ann.	Cap.	Ann.
Slag	-	0.39%	-	0.39%	-	0.56%
Total	-	0.39%	-	0.39%	-	0.56%

SECTION IV

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